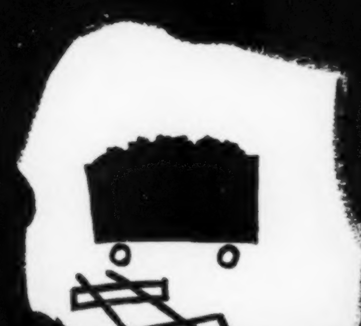
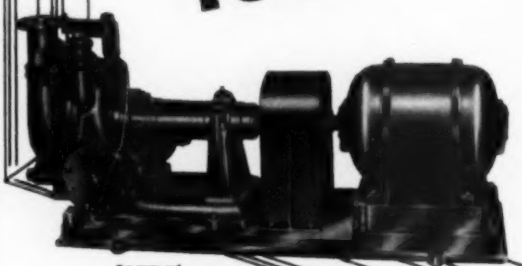
MAY 1952

Mining in Mexico



WILFLEY
centrifugal PUMPS

Handling rod mill discharge



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The Model "K" WILFLEY is THE pump to buy when the job calls for a rugged, heavy-duty pump that delivers higher efficiencies, stepped-up production, worthwhile power savings and lower operating costs. Continuous, 24-hour-a-day performance. Easy interchangeability of wear parts. Low maintenance costs. Every application individually engineered. Write or wire for details

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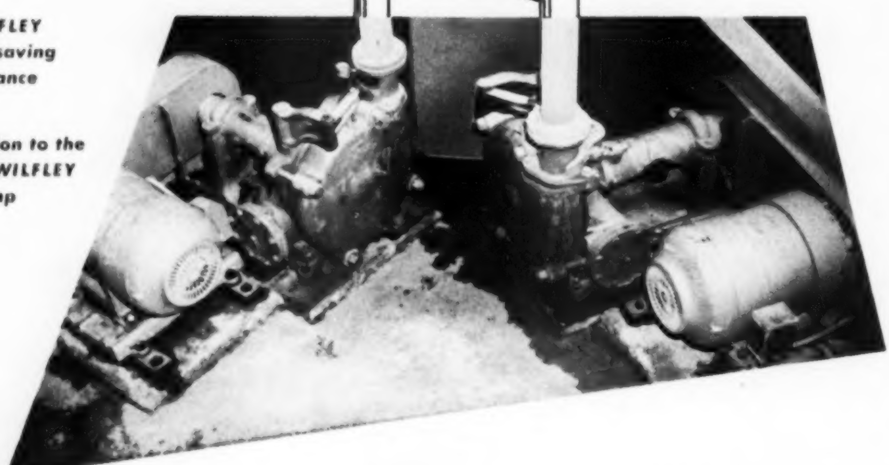

5" Model "K" Wilfley Sand Pumps in a large western sand and gravel plant handling 30% solid discharge containing highly abrasive 3/16" silica from rod mill.

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 4 NO. 5 MAY, 1952

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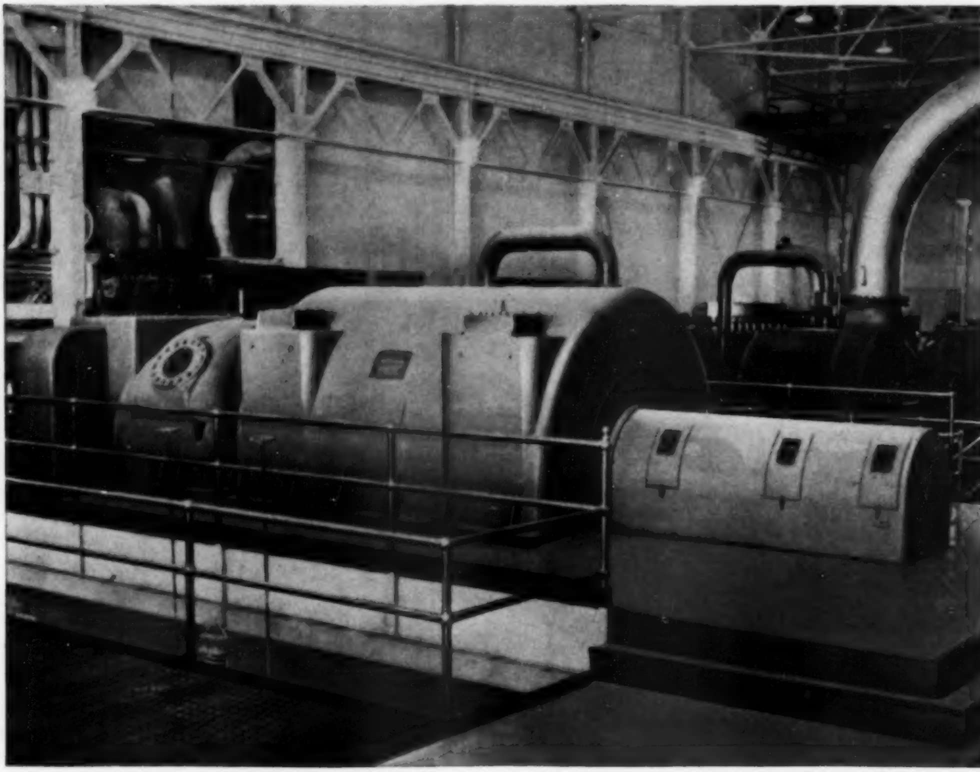
COVER

A Toltec idol symbolizes the ancient civilization of Mexico that first opened some of the mines of importance today.

FEATURES

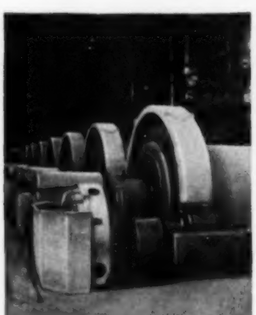
ARTICLES

TRANSACTIONS

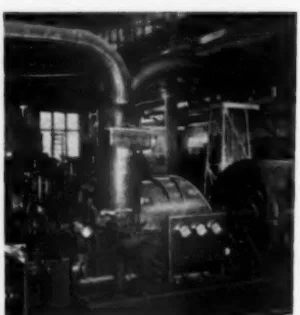


1 **NEW G-E STEAM TURBINE-GENERATOR** generates high-voltage power for this copper company's concentrating and smelting plant. Rated at 10,000 kw, 3600 rpm, the single-stage unit replaces an older 6000-kw unit which generated at 480 volts. Like every G-E turbine-generator, it is custom-built from standard components to meet specific operating conditions.

Power system modernized for

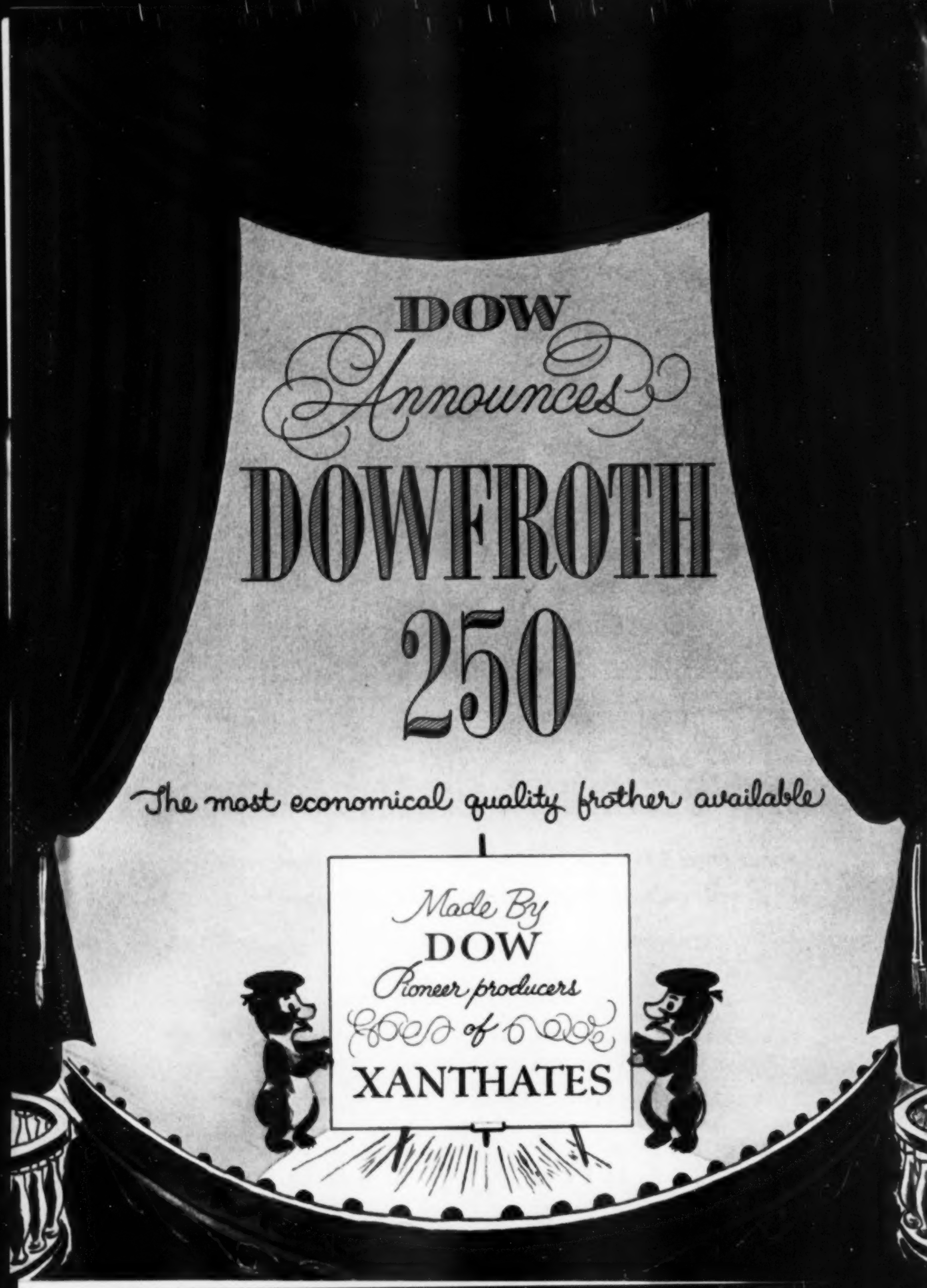


POWER-FACTOR IMPROVEMENT is provided by 28 200-hp synchronous motors driving plant's ball mills.



MECHANICAL POWER to drive two turbo-blowers is generated from process steam by 1915-hp G-E mechanical-drive turbines.

LOW-COST HAULAGE from the copper plant's open pit mine 15 miles away is provided by these two G-E 750-volt 85-ton electric locomotives.



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Letters to the Editor

Principal Cobalt Source

My bets are on you, every time! But who is right? In the "cobalt issue" of our favorite magazine, January 1951, you stated: "By far the best immediate United States prospect for large amounts of cobalt is the Blackbird mine of the Calera Mining Co., which is operating for Howe Sound Co., in Leinhi County, Idaho." In the March 1 issue of *Saturday Evening Post*, Ernest F. Mechlin, Jr., chief cobalt section, National Production Authority, states: "The Bethlehem Steel Co.'s Cornwall, Pa. ore bed is the principal domestic source."

Maybe he should have said: "*was* the principal domestic source", or perhaps Howe Sound's Idaho cobalt is still in the prospect stage in Mr. Mechlin's opinion, perhaps in yours too.

KATHARINE W. CARMAN
GEOLOGIST
933 MICHIGAN AVE.
EVANSTON, ILLINOIS

Mr. Mechlin is correct in his statement that Cornwall is the principal domestic source of cobalt, but within the year, Blackbird production should greatly exceed that from Cornwall. Calera is expected to ultimately produce 3 million lb per year, whereas Cornwall averages 520,000 lb per year. —Editor.

Drill Testing Paper Questioned

The paper presented at the Annual Meeting by Paul Russell is significant in that it appears that holding rods in a hardening furnace for a longer-than-usual period will not adversely affect the life of a drill rod. It is known that unless the drill rod is heated and quenched in its full length, a transition zone is always obtained. Proper heat treatment can minimize undesirable change in hardness in the treated end of a rod.

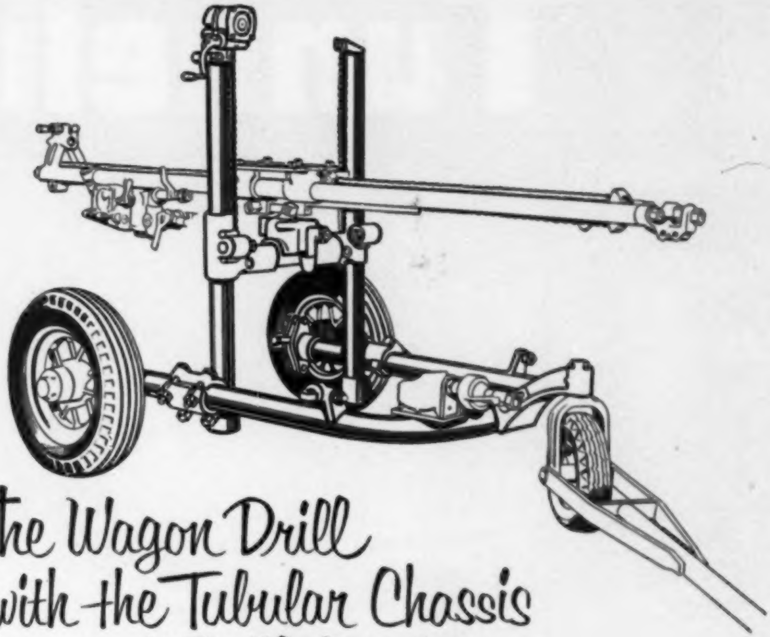
The test described by the paper was made to compare rods with the metallurgical notch (abrupt hardness change) emphasized by a long heat in one heat and and the metallurgical notch minimized in the other—the standard heat. The heat treatment may not have accomplished this difference. It may have been caused by the intermittent shooting of flame out of the hardening furnace door. It would be interesting to learn the effect of the long heat and the standard heat by making hardness surveys on a number of the rods.

The results are especially interesting in view of the fact that the long heat rods produced surprisingly more satisfactory results. This stresses the need for hardness surveys to determine what the long and standard heats really produced in the rods. There seems to be a great deal of difference in the penetration rate. Suspicion arises as to variations in drilling conditions in respect to sharpness of the bits or differences in the rock drilled. These factors cannot be overlooked in final evaluation of the work.

In connection with chemical variations due to incidentals including Ni and Cr, the Shepherd PF hardenability tests offer a better basis for comparison. Hardenability tests compare the overall effect of all elements. The chemical differences do not explain the variations in drilling performance.

Another interesting point is the location of the breaks. It is normal to have all the breaks in the forged, heat treated ends. Failed rods should be examined to determine if rough or rusted surfaces account for some of the failures in the shank or thread ends.

W. F. TOWNE
BETHLEHEM STEEL CO.



The Wagon Drill with the Tubular Chassis and Drill Carriage

G-300 Wagon Drill

Write for further information.

The rugged tubular chassis and drill carriage of the G-300 WAGON DRILL provides rigidity and strength without unnecessary weight.

The tubular "H" structure that supports the drill carriage gives greater stability than the conventional design and minimizes vibration. The all-around sturdy construction of the G-300 maintains correct alignment at all times, insuring maximum drilling efficiency with minimum upkeep.

G-300 Wagon Drill

- is designed to take full advantage of the high drilling speed and strong rotation of the CP 4-inch 70-N Drifter.
- has controls conveniently centralized.
- is equipped with a specially designed feed motor and cone-gear drive for greater load-carrying capacity.
- has a heavy-duty centralizer.
- adaptable for all types of wagon drill work. Wheels can be turned at right angles to facilitate line drilling or drilling close to a ledge or wall.

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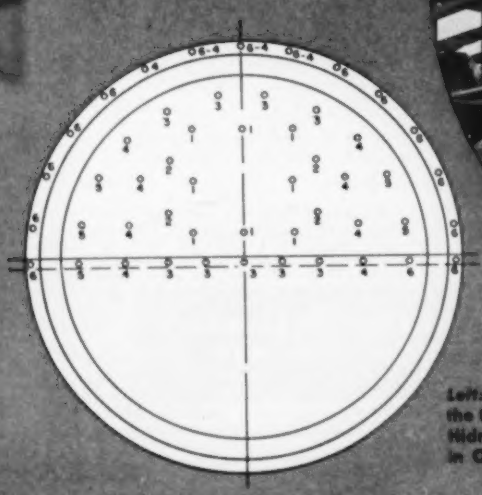
Tunneling

Right: Six-drill jumbo with Le Roi-CLEVELAND power feed drifters and air-motor booms on a 1½-ton truck. 1¼" round-lug steel and 2" carbide bits. Air supply — two 500 cfm compressors.



Left: Standard drill pattern for 30-hole round. Typical of the top-heading and bench method used by Construcciones Hidraulicas, S.A. in driving three 27'-bore circular tunnels in Obregon, Sonora, Mexico.

MEET THE AUTHORS

B. S. Crocker *(Screened Ore Used for Fine Grinding at Lake Shore Mines, P. 499)* was born in Toronto, Ont. and attended Saint Andrews College and University of Toronto. He received a B.A. and M.A. degree. He was graduated with honors and received a fellowship in metallurgy. Mr. Crocker did post-graduate research at University of Toronto from 1933 to 1934. He was employed by Beattie Gold Mines in 1933. He did research in milling at Lake Shore Mines from 1934 to 1936 and from 1936 to 1940 was head of experiment laboratory. At the present time he is mill superintendent with Lake Shore Mines at Kirkland Lake, Ont., Canada. Grinding, cyanidation, flotation and roasting are of special interest to him. He is a member of AIME. Mr. Crocker has presented several papers before C.I.M.M. and also holds membership in this society as well as Professional Engineers of Ontario. Skiing, yacht racing and photography are his favorite hobbies.

R. D. PARKS

R. D. Parks *(Chromium, P. 469)* was born in Lake Linden, Mich., and attended Lake Linden high school and Michigan College of Mines in Houghton, Mich. He received a B.S., E.M., and M.S. degree. From 1938 to 1940 he was an associate professor of mining engineering at Michigan College of Mining and Technology. During World War II period he was granted a leave of absence from Massachusetts Institute of Technology and was with the War Production Board in Washington, D. C. Since 1940 he has been an associate professor of mineral industry with Massachusetts Institute of Technology. Since 1950 he has also been an associate consultant with Behre Dolbear & Co., New York and since 1949, at times, he has been consulting mining engineer with U. S. Bureau of Mines in Washington, D. C. An AIME member, he has presented previous papers before the society—TP #708, "Total Profits vs. Present Value in Mining" and TP #1883, "Corundum—A Vital Wartime Abrasive". Mr. Parks is author of text and handbook "Examination and Valuation of Mineral Property", 3rd edition, 1949, Addison Wesley Press, Cambridge, Mass. Photography and stamp collecting are his favorite pastimes. He now resides at 3 Upland Road, Lexington 73, Mass.

F. C. Bond *(The Third Theory of Comminution, P. 484)* attended University of Denver and Colorado School of Mines. He received an E.M. in 1922 and M.S. degree in 1926. He was an assayer and millman with New York and Honduras Rosario Mining Co. Mr. Bond was employed by Tennessee Copper Co. from 1929 to 1930. He has been with Basic Industries Research Laboratory, Allis-Chalmers Mfg. Co. since 1930. An AIME member, he has presented various papers before the society—"Grindability of Various Ores" and "Crushing Characteristics as Determined from Screen Analyses" are but a few of papers he has presented on subjects of crushing, grinding and mineral dressing. He holds membership in American Chemical Society, C.I.M.M. Milwaukee Engineering Society, and Astronomical Society of the Pacific. Mathematics and astronomy are his favorite pastimes.

MEET THE AUTHORS Cont'd

R. R. BRYAN

R. R. Bryan *(Real del Monte Finds, P. 464)* was born in Denver, Colo. and attended East Denver high school and Colorado School of Mines. Mr. Bryan was a testing engineer with Anaconda Copper Co. He was chief chemist with Utah Copper Co. He was employed by Sunnyside Mining & Milling Co. as metallurgist. Mr. Bryan has been a metallurgist, assistant general superintendent of mills and general superintendent of mills at Cia. de Real del Monte y Pachuca. He presented a joint paper with Mr. M. H. Kuryla, "Milling and Cyanidation at Pachuca", Transactions, AIME Volume 112. He resides in Pachuca, Hgo., Mexico. Photography is his favorite hobby.

E. L. OHLE

E. L. Ohle *(Geology of the Hayden Creek Lead Mine, P. 477)* was born in St. Louis, Mo. and attended University City high school. He attended Washington University in St. Louis and Harvard University. His honors included a Phi Beta Kappa, Sigma Xi and Omicron Delta Kappa. He was employed by American Zinc Co., Mascot, Tenn., Batesville, Ark., and Joplin, Mo. as geologist and assistant mine superintendent. He was a geologist with St. Joseph Lead Co. in Bonne Terre, Mo. Problems of special interest to Mr. Ohle are geology of lead and zinc deposits in limestone. An AIME member, he presented a previous paper before the society with Mr. Hugh McKinstry entitled, "Ribbon Structure in Gold-Quartz Veins." Mr. Ohle holds membership in Geological Society of America, Society of Economic Geologists and Phi Delta Theta. Sports in general are his favorite pastimes.

C. L. Boeke *(Deleading Zinc Concentrate at the Parral and Santa Barbara Mines—co-author with G. G. Gunther, P. 485)* was born in Lena, Ill. and attended Lena high school and Colorado School of Mines. He received an E.M. degree in 1920. Mr. Boeke has been with the American Smelting & Refining Co. in Mexico for a great number of years. Minerals beneficiation and practical mill operating problems are of special interest to him. Mr. Boeke is an AIME member.

W. P. Hewitt *(co-author with C. M. Signer "San Antonio Mine—Landmark on the Path of the Conquistadores", P. 459)* was born in Manila, P. I., and attended Yonkers high school and Columbia University. He received B.A., B.S., E.M. and Ph.D. degrees. Mr. Hewitt has been with American Smelting & Refining Co., Mexican Div., Geology Dept. from 1933 up to present time. An AIME member, he resides in Chihuahua, Mex.

G. G. Gunther *(co-author with C. L. Boeke, P. 485)* was born in Germany and attended the University of Freiberg, Saxony, Germany. He received a mining engineering degree in 1926. From 1925 to 1927 he was a coal miner in Germany and Pittsburgh, Penna. Mr. Gunther worked as junior engineer, mining and milling from 1927 to 1929 at Kirkland Lake, Ont. He worked in El Paso, Texas and at various American Smelting & Refining units in Mexico. At present he is mill superintendent for this company at Santa Barbara, Chih., Mexico.

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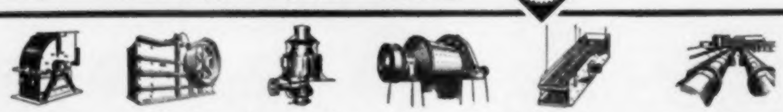
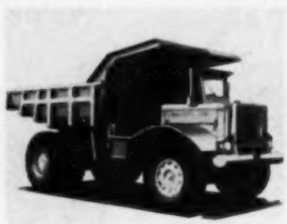


Pulverator | Jaw Crushers | Gyratory Crushers | Grinding Mills | Vibrating Screens | Kilns, Coolers, Dryers

Dump Truck

Kenworth Motor Truck Corp. is beginning full scale production of its new heavy-duty, end-dump earth mover, model 801, a truck engineered for earth moving under the most adverse operating conditions. Payload capacity of the Kenworth earthmover is 30,000 lb and it is over-tired for safety, flotation and high tire life. The truck's body capacity, struck measure, is 9.9 cu yd, with heaped



load at 11.9 cu yd. The earth-mover has a full anti-friction bearing mounted, power assisted steering gear, simplified controls, minimum turning radius and wide axle track to insure ease of handling, maximum maneuverability and high stability. The offset cab of the Kenworth visibility for the driver, before and after. **Circle No. 1**

Safety Equipment

A compact, safe, and efficient valve assembly approved by the United States Bureau of Mines for use with MSA air line respirators, abrasive masks, and abrasive helmets has been developed by *Mine Safety Appliances Co.* The new MSA continuous air flow control valve regulates air flow from a compressed air source to an operator using respiratory protective air line equipment. Three parts—a centridge container, a cartridge and the air flow check valve—comprise the entire assembly of the control valve. A position action adjustment knob on the cartridge container regulates the amount of air flowing to the operator. The valve can be detached from the air line instantly by pressing the trigger on the cartridge container near the check valve. As the trigger detaches the hose, the spring loaded ball-type valve, which is attached to the end of the hose, automatically stops the flow of air. **Circle No. 2**

Belt Trainer

This device automatically keeps wandering conveyor belts aligned and can be installed on any make of conveyor which has the return belt exposed beneath the bed. It is designed for use on fabric or rubber-covered belts 3/16-in. or more thick, operating at speeds up to 200 fpm. The trainer does not interfere with normal conveyor operation, and does not mark or wear the belt in any way. In use the trainer is bolted to the flanged rails on the underside of the conveyor. Flat sides of the belt run between two sets of knurled spring loaded rollers, with the belt edges contacting two sets of equalizer posts. Manufactured by *The Rapids-Standard Co., Inc.* **Circle No. 3**

Scrapers

Caterpillar Tractor Co. announces a pair of new scrapers for use with the Cat DW10 tractor. With this choice, the equipment user can match his rig more closely to job requirements. The new Cat No. 10 scraper is somewhat lighter than before, with capacity of 7 cu yd struck and 9 cu yd heaped. Top extensions (sideboards) may be attached to

either scraper for increased capacity where the material does not exceed a weight of 2800 lb cu yd. The scrapers are similar in basic design. Both have a flat, double-bottom bowl of high-tensile steel. A stinger blade with reversible cutting edge is standard equipment. Cable rigging provides for positive loading and ejection. The wheels turn on tapered roller bearings. Air brakes are synchronized with the tractor brakes. **Circle No. 4**

Trailer

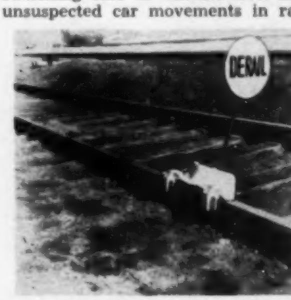
The Phillips Trail-Skid, a two-wheel trailer primarily designed for handling long lengths of materials in confined spaces, is a recent development of *Phillips Mine & Supply Co.* Pulled by either fork or platform lift truck, this new unit also is adaptable for transporting multiple pelletized loads and for general plant hauling. In operation, a lift truck engages the coupling mechanism mounted on an articulated turntable, lifts the skid legs free of the ground and moves the unit on its rear wheels. Cranes or lift trucks may be used to load and unload the Trail-Skid, which can be used either with or without stakes. The unit is available in five models, ranging in length from 7 to 16 ft. All models have a capacity of 10,000 lb, and feature solid rubber-tired wheels with Timken roller bearings. **Circle No. 5**

Vibrator

A new and larger vibrator has been added to the patented Peterson "Vibrolator" line manufactured by *Martin Engineering Co.* The new model, specified as the DV 51, is designed to tackle the feeding problems that arise when larger hoppers and bins are necessary in materials handling. The DV 51 gives an all-directional vibration that effectively moves materials without damage to the faces of hoppers or bins. It has a 2-in. ball that weighs one pound—an increase of 66 pct from the largest model previously manufactured by Martin. The new "Vibrolator" is easily installed and requires practically no maintenance. It is being used successfully in the movement of all types of materials that must be fed from hoppers and bins including sand, gravel, coal, and many other materials that resist movement toward hopper outlets. **Circle No. 6**

Derail Unit

Availability of a portable derail unit for temporary derailing service and safety is announced by *The Nolan Co.* The Nolan derail is recommended for use wherever workers may be endangered by wild cars, switching cars or uncontrolled and unsuspected car movements in rail



yards, spurs, etc. This derail is particularly adapted to mine use as protection at room necks, entries, etc. Derailment is accomplished in either direction, on right or left hand rail. **Circle No. 7**

Speed Reducer

A new double reduction shaft-mounted speed reducer with capacity to 43 hp, and for output speeds from 12 to 110 rpm, has been announced by the *Dodge Mfg. Corp.* The new No. 7 reducer greatly increases the applications for Dodge torque-arm speed reducers. It has 59 pct greater horsepower capacity than the No. 6. Like previously announced models the No. 7 is shaft-mounted and anchored with a torque-arm which fastens to any fixed object. **Circle No. 8**

Free Literature

(9) CRUSHERS: *Stephens-Adamson Mfg. Co.* has just released a new bulletin covering their complete line of crushers. Included are Knittel ring-type crushers, both single and double rotor: single roll crushers: double roll crushers and two-stage, double-roll crushers. These units are designed for reducing coal, coke, glass, cullet, gypsum, line, metal turnings, sugar, sulphur and other lump materials. Latest addition to the S-A line is the double rotor Knittel crusher for damp coal and similar sticky materials. In single rotor, ring-type units there is a tendency for sticky material to build up on breaker plates, greatly reducing capacity and often completely choking the feed. The double rotor unit eliminates breaker plates entirely and is able to handle the same tonnage per hp of wet coal as the single rotor unit handles dry.

(10) GRINDING MILLS: Comprehensive engineering data on grinding mills for rock products, cement, and mining industries is contained in a new 44-pg bulletin released by *Allis-Chalmers Mfg. Co.* Generously illustrated, the bulletin suggests points to consider when choosing a grinding mill, gives the basic design of the different types of mills, describes grinding circuits, and furnishes equations and tables for use as a guide for mill selection, along with closed and open circuit grindability indices unobtainable from any other source.

(11) PRODUCT CATALOG: *The Bin-Dicator Co.* has just issued a catalog describing and illustrating the company's products. This company markets a complete line of bin level indicators which are widely used throughout industry to indicate the level of granular and pulverized materials stored in tanks, silos, hoppers and bins. These units also actuate various types of signals, if required, and can be used to start and stop loading and filling machinery, as required by the level of content in the storage unit. The new catalog supplies complete installation data for the various types of units; for thin or thick walled bins, for inside or outside location, and for suspended interior installation.

(12) BUCKET TROLLEYS: Users of overhead or bridge type cranes with 4-rope bucket trolleys will be interested in a new 44-page bulletin #2392 *Blaw-Knox Co.* has just published on 4-rope buckets. There are numerous illustrations of bulk material handling including coal, ores, chemicals, etc., with performance data and comparisons of cargo dispatch and discharging vessels, barges and railroad cards. To provide further assistance to engineers and users, helpful information and diagrams are presented on 4-rope bucket types, reeving, arrangement of cables for various types of cranes, cable life as influenced by sheave diameter, and how to determine the increase in payload through use of anti-friction bearings in bucket sheaves.

(13) GAS, AIR, TURBINES: *The Pyle-National Co.* has just released six new bulletins describing the company's line of high-efficiency impulse type steam, gas or air turbines, for mechanical drive, which has recently been expanded for general industrial use. One of the bulletins is devoted to each of the five types of turbines available, ranging from ¼ to 120 hp, giving performance charts, construction details, dimensions and weights and selection data. The sixth bulletin provides general turbine information.

(14) SHAFT UNIT DRIVES: *Elliott Mfg. Co.*, has come out with a complete new line of ready-to-use Econoflex Flexible shaft unit drives, available for shipment from stock, in four size ranges—heavy duty, medium duty, light duty and drill shaft unit. The heavy duty range provides 180 different standard units, varying in style of end fittings, bearings, lengths of shafting,—in core diameters from ½ in. to 1¼ in. The heavy duty drives are for applications requiring high strength, moderate flexibility, and lower operating speeds, as for power take-offs. The Econoflex medium duty drives are available in 6 core sizes, from 5/16 in. to 5/8 in. diameter. They may be attached to any power source—electric motor, gasoline engine, drill press spindle or countershaft and may be used to perform all the operations which are possible with a complete flexible shaft, machine, such as grinding, wire brushing and drilling.

(15) TRUCK BULLETIN: Bulletin #1613 issued by Baker Industrial Truck Division of the *Baker Raulang Co.* describes model CXB and CXF locomotive type crane trucks. Model CXB has a rated capacity of 6000 lb at 7-ft radius. Model CXF's rated capacity is 10,000 lb at 5½-ft radius. The new six-page bulletin uses cutaway photos of component parts to make construction details easy to follow. Complete engineering specifications and drawings are also included, along with action photographs that show these trucks at work in a variety of applications.

(16) CONVEYORS: A new 24-page illustrated Book No. 2444 on positive action oscillating conveyors for conveying, feeding, cooling, screening a great variety of loose belt materials, is announced by *Link-Belt Co.* The book includes dimensions, weights, and capacity and horsepower charts on Torsion Mount oscillators for heavy duty as well as for the recently announced Flexmount oscillator for the lighter-duty applications. In these machines, a one-piece metal trough mounted on supports is given an upward and forward oscillating motion by an eccentric drive.

(17) MICROSCOPES: These microscopes are of entirely new and of an improved design, featuring a patented trip-in Bertrand lens and a revised substage construction which permits full field illumination at all powers without complicated condenser fittings. The polarizing microscope is used in research and control operations in the fields of mining technology, and the metals. *Cooke Troughton & Simms.*

Mining Engineering — **May**
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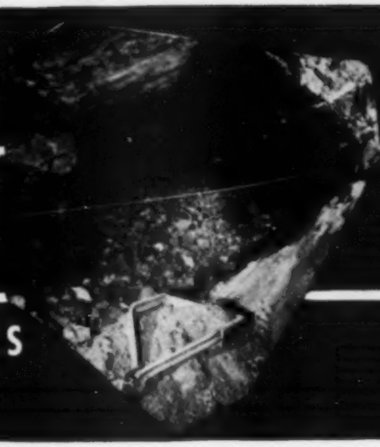

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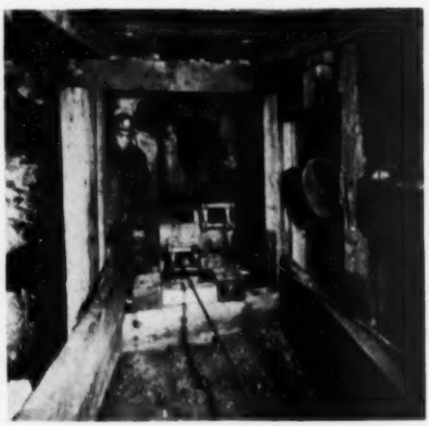
AIR SLUSHER



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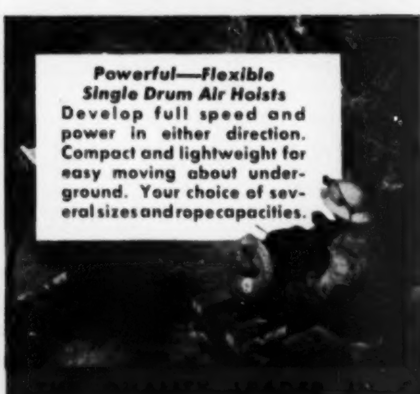

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Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1**. Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Graduate Mining Engineer, 31, married. Six years' engineering experience, mining and construction, U. S. A. and Alaska. Underground metal mining, production, development, geology, mine engineering. Engineering on airfield construction, earthwork, location work, mapping, topography, road location, office work. Commercial pilot license. Desires position mining or construction. Available 45 days. Prefer Latin America. M-676-477-E-8—San Francisco.

Mining Engineer, 42, married, one child. Fourteen years' mining with much supervisory experience in developing and operating metal mines; also mine safety engineering. Familiar with many large and small mines in the western U. S. and Mexico. Excellent references. M-677-345-E-1—San Francisco.

POSITIONS OPEN

General Manager, 35 to 45, with at least ten years' mining or construction, plant engineering and industrial management experience in Latin America. Salary, $12,000 to $15,000 a year plus bonus. Location, South America. Y6882.

Open-Pit Superintendent, mining graduate, with at least ten years' open-pit mining and stripping experience, including truck operations, to take charge of manganese project. Salary, $12,000 to $15,000 a year plus bonus. Location, Southwest. Y6844.

Consulting Mining Supervisor who has had considerable experience in the actual operation of metal mines, particularly tin and tungsten. Considerable traveling. Salary, $15,000 to $18,000 a year. Location, New York, N. Y. Y6822.

Engineers. (a) Chief Safety Engineer, 35 to 50, mining graduate, with considerable safety experience covering hard rock mining, milling, metallurgical, general maintenance and power operations, to direct safety staff, supervise industrial hygiene, prepare manuals, etc. Salary, $12,000 a year. (b) Chief Warehouse Superintendent, 35 to 50, with at least ten years' industrial and mining equipment experience, to supervise inventory records, schedule purchases, plan economical handling of receiving, shipping of machinery, tools, parts, materials, foods, consumer goods, etc. Salary, $8000 to $10,000 a year. Must speak Spanish. Location, South America. Y6784.

Engineers. (a) Mine Foreman with at least three years' underground experience for lead-zinc operation. Salary, $4800 a year plus housing and family accommodations. (b) Shift Boss, preferably engineering graduate, with underground experience. Salary, $4200 a year plus housing and family accommodations. Must speak Spanish. Location, Peru. Y6679.

Mining Engineer to take care of management and supervision of four small mines running from 75 tons per day up to 400 tons a day. Must be able to speak Spanish and get along well with native help. Salary open. Location, South America. Y6590.

Geologist, experienced in structural interpretation, routine underground and surface mapping for gold vein mine. Salary, $6000 a year plus bonus of one month's salary. Knowledge of Spanish desirable. Location, Colombia, S. A. Y6581.

Mining or Metallurgical Engineer, with degree in mineral dressing engineering, not over 28, preferably with one or two years' experience in the milling of metals for sales contact work. Prefer single man. Salary, $4800 to $5400 a year. Location, New York, N. Y. Y6494.

Research Engineer, 25 to 40, graduate in mining or metallurgical engineering, preferably with a Master's degree in mineral dressing, with at least three years' experience in ore dressing, research or mill work, preferably with experience in nonmetallic flotation. Location, Florida. Y6465.

Engineers. (a) Assistant Chief Engineer with underground experience and preferably a working knowledge of Spanish. Salary, $4200 a year plus bonus. (b) Mill Superintendent with broad experience in milling. Should have working knowledge of Spanish. Salary, $5400 a year plus bonus. (c) Diamond Drill Foreman with knowledge of Spanish. Salary, $6000 a year. Single status for three months. (d) Metallurgist to conduct and supervise ore dressing, laboratory experimenting and concentration of tin ores by gravity, flotation and volatilization technique. Three year standard contract with three weeks vacation each year, plus living quarters, medical and hospital service. Transportation to Bolivia for applicant, wife and one child paid. Y6402.

Mill Superintendent with mining, crushing and flotation experience, to take charge of equipment operation project covering mining, concentrating by flotation, drying, grinding, calcining, magnetic separation, etc. Salary, $5000 to $6000 a year. Location, South. Y6348.

Mining Consultant to estimate cost of sinking shaft and bringing mine into production, for barrelled mica pegmatite operation. Salary open. Location, New York, N. Y. Y6272.

Junior Mining Engineer with some operating and exploration experience, for clay mining project. Salary open. Location, East. Y6185.

Assistant Geologist for work on open-pit mine maps and sections, field supervision of ore drilling and sampling program and ore property examinations. Salary to start, $3600 to $3900 a year. Location, Texas. Y6172.

NICKEL ALLOY IRONS
develop improved properties
plus all the basic advantages of plain cast iron

PLAIN GRAY IRON is, structurally, a steel matrix containing graphite flakes. Engineering, physical, processing and service properties are wholly dependent upon the character and disposition of these flakes, and upon the nature of the matrix.

The matrix of nickel alloyed irons closely resembles the pearlitic matrix found in high carbon steels, whereas the matrix of ordinary plain iron resembles that found in low carbon steels. Compositions of nickel alloy irons can be adjusted to reduce "chill" in thin sections without risk of forming "spongy" regions in heavy sections. This promotes uniform strength, improved machinability, pressure tightness and wear resistance.

Hardness in nickel cast irons results from improvement of the matrix. Chilled areas and hard carbides, which impair machinability, are obviated. Nickel improves response to heat treating. In fact, use of nickel alone or with other alloying elements plays an important part in meeting a variety of requirements.

Accordingly . . . nickel alloyed irons permit production of castings with high levels of the following properties:

Strength

Tensile and transverse strengths of castings are greatly increased by the addition of nickel to cast irons of properly adjusted base mixture. The ratio of compressive strength to tensile strength is retained. Greater uniformity of strength in thick and thin sections is achieved.

The International Nickel Company, Inc.
Dept. ME, 67 Wall St., New York 5, N. Y.
Please send me booklet entitled, "Guide to the Selection of Engineering Cast Irons."

Name ______________ *Title* ______________

Company ______________

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EMBLEM OF INCO SERVICE

Elasticity

The elastic modulus increases with strength. In this respect nickel-containing irons of the high strength type possess good stiffness and do not deform permanently under loads that would be damaging to irons of lower elastic modulus.

Damping Capacity

The damping capacity inherent in gray cast iron is not impaired by the presence of nickel.

Wear Resistance

The uniformly pearlitic matrix of nickel cast irons appreciably improves wear resistance. The uniformly fine graphite flake distribution, achieved in suitably processed irons *without formation of a poor wearing dendritic condition,* affords optimum resistance to wear and galling.

Pressure Tightness

Characterized by dense grain structure and fine dispersion of graphite throughout, nickel alloy irons are close-grained and offer an extraordinary degree of pressure tightness under high hydrostatic pressures, without sacrificing machinability.

Applications

Heavy machinery frames and *beds* are typical of cast parts that benefit from the rigidity and good damping capacity of nickel cast irons. *Cylinder* and *pump liners, gears, dies, machine tool ways, saddles* and *tables* exemplify parts produced in nickel irons to assure greatly increased strength and wear resistance. And nickel alloyed iron is used for *heavy duty brake drums* to resist heat checking, thermal shock, wear and galling. The nickel cast irons are readily heat treated, and respond particularly well to flame and induction hardening.

At the present time, the bulk of the nickel produced is being diverted to defense. Through application to the appropriate authorities, nickel is obtainable for the production of engineering nickel cast irons for many end uses in defense and defense supporting industries.

THE INTERNATIONAL NICKEL COMPANY, INC. 67 WALL STREET NEW YORK 5, N. Y.

Restrictions on the use of zinc will be a thing of the past in a few months. Government allocations on the metal are also expected to go by the board. Concurrently, action is also expected in reference to lead allocations. Government stockpile goals for zinc have been reached.

The International Union of Mine, Mill and Smelter Workers lent support to United Steel Worker demands and at the same time put in their own bid for a 25-cent an hour increase at a meeting held in Denver. While backing the USW, the IUMMSW charged the Steel Workers with membership raiding.

Iron and steel companies producing about 92 pct of steel ingots in the United States, showed nearly a 13 pct decline in net profit during 1951, compared with the previous year, despite a 22 pct increase in their combined revenue. Federal income tax rose 61 pct, to more than $12 million.

New iron ore reserves are being sought in Canada by Jones & Laughlin Steel Corp. The company has acquired exploration options near Petersborough on the north shore of Lake Ontario in Durham, Northumberland and Frontenac Counties. A fourth property is southwest of Ottawa, in the Kingston, Ontario area.

The National Production Authority expects to fill all requests for copper filed under the controlled material plan for the second quarter. While defense demands are still taking big chunks out of the supply, the worst of the shortage of the metal for civilian use seems to have passed. Brass demands are on the increase with military orders the major factor.

An entirely new area for uranium prospecting is opening up in South Dakota and Wyoming. Carnotite was found last June in the Craven Canyon area about eight miles north of Edgemont, S. Dak. The Department of Interior is also making a large block of public land in the Pumpkin Buttes area available for further exploration, following discovery of significant deposits of uranium.

Venezuelan Ministry of Mines Geologists have discovered a bauxite deposit estimated at about 2 million tons in the Uriman region north of Upata in the state of Bolivar.

The Atomic Energy Commission has been investigating several sites in the Ohio valley with the idea of constructing a $1 billion plant. The new operation will be part of the President's $5 to $6 billion atomic expansion program. The plant will be used to separate two types of uranium gas and will take about four years to build.

The Dominion Steel and Coal Corp., Ltd., is using the Dosco Miner to cut and load coal from the seam without employing explosives. The machine cuts and loads coal concurrently.


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TO DIAMONDS IN SOUTH AFRICA . . .
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The world-wide use and acceptance of "SYMONS" Cone Crushers by leading companies engaged in the production of ores and industrial minerals is evidence that the "SYMONS" Cone is outstanding in its performance from the standpoint of finely crushed product—great capacity and low crushing cost.
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Smelter Nears Completion

New zinc fuming plant of the Cia. Metalurgica Del Norte S.A., a subsidiary of the American Smelting & Refining Co., nears completion at the Avalos Smelter, near Chihuahua City, State of Chihuahua, Mexico. The plant is scheduled for completion on July 1, 1952. The erection contractor is Utah Construction Co. Supervising engineer for Asarco is Leslie E. Harris, AIME.

U. S., Britain Lend Financial Aid to South African Gold Miners For Uranium Production

South African gold mines are expected to begin producing uranium next month with the financial aid of the United States and Great Britain, according to South African Minister of Mines J. H. Viljoen.

Viljoen predicted that within three years South Africa will be one of the world's most important uranium producers. The West Rand Consolidated mine will be the first to start producing uranium, with a dozen more mines scheduled to follow suit. The mines will be strung out 200 miles from Johannesburg to the Orange Free State's gold fields in order to minimize the effect of air attack.

Britain has set aside about $1.48 million, with the United States contribution at least equaling that since 1951. The two sponsoring nations have agreed to supply money for the purchase of needed equipment. A three-power agreement binds the nations to secrecy concerning the process and production figures.

Small deposits of thorium—potential atomic energy source—have been discovered in the Transvaal, Viljoen said.

Chilean Mine Strike Endangers Copper Supply

United States supplies of Chilean copper, already endangered by a demand for a higher price by the South American country, is also threatened by a possible strike by miners for higher wages.

Some 10,000 workers at two big Anaconda Copper Mining Co. mines threatened to walk out unless wage demands are met. The two properties are the Porterillos mine, with an annual production of 50,000 tons, and the Chuquicamata, with an annual output of 172,500 tons.

An Anaconda official said, the increase, if granted, would cost the company about 1.5 billion pesos. The added costs would put the firm out of business. Copper workers are the best paid in the country, he added, and they have not felt the inflation as severely as others because prices at copper plants are frozen.

Thus far, DMPA-Chilean talks on copper prices have brought no results. The talks are reported to center around Chilean demand for 30 cents per lb, but the report lacks official confirmation. It is also reported that the conversations concern only about 20 pct of production, hitherto sold on the European market.

Last May Chile agreed to sell 80 pct of its production to the United States at 27½ cents per lb. The rest Chile could sell on the world market. The price last year averaged out to about 54 cents per pound. A resultant 33½ cents per lb average for total production was realized by Chile. World copper prices have declined in recent months. Chile is now reported willing to sell the 20 pct reserved for world trade to the United States for 33½ cents, rather than run the risk of further market declines. The price would then average out to 30 cents per pound.

Expect New Jersey Zinc Co. Mine Will Add Strength to United States Production Leadership

Any danger to United States zinc production leadership is expected to be headed off by the coming of the New Jersey Zinc mine and mill, four miles south of Bethlehem, Pa. It is expected that the new operation will offset loss of production from New Jersey's Franklin Furnace mine, due to be closed soon because of ore depletion.

Installation at Friedenville, started in the last half of 1947, has been slowed by mishaps and other delays. Sinking of the 1250-ft shaft by the E. J. Longyear Co. was held up by ground water. Saucon Valley mines were closed in 1892 because of that handicap.

Clay-filled water bearing cavities resulted from ground water percolation which dissolved the limestone and dolomite. In sinking the new shaft it was decided to use a pre-grouting method for protection against the water, but clay filling in the openings prevented proper sealing with the grout.

Despite losing the shaft through flooding several times, progress has been good. The shaft is now at 1200 ft, with the crusher station at 1170 level under excavation. Access to the mine will be 13x20 ft, three-compartment shaft, timbered with steel sets on seven-ft centers and concreted for the full length.

The mill and services buildings, main surface plant structures were designed to conform with the existing community. The services building, almost completed, contains the hoist and compressor rooms, mine shops, change house, and administrative offices. Mill construction is scheduled to start this year.

The sphalerite orebody is in Ordovician age limestone. Zinc sulphide or sphalerite is the only value contained in the ore. Production from the relatively low grade deposit is scheduled to start sometime in 1953.

Ford Ore Fleet to Get Largest Addition in 1953

The third and largest ore carrying vessel in the Ford Motor Co. fleet is scheduled to join two other sister ships at the start of the 1953 season. Its keel has been laid at the River Rouge yards of the Great Lakes Engineering Works.

The ship will be named the William Clay Ford for the youngest of the Ford brothers. It will be 647 ft long, with a beam of 70 ft. Capacity will be 19,000 tons. A speed of 16 miles per hour will be possible with the 7000 hp oil-fired turbine.

The other Ford ships, in service on the lakes since 1924, are 612 ft long, and are diesel powered.

Bolivian Demands Scored

The chairman of the Senate Preparedness Subcommittee voiced opposition to the demand for a higher price for tin made by Bolivia. Senator Lyndon Johnson (Dem. — Tex.) pointed to recent contracts with other countries as examples of "straightforward dealings."

In a letter to Harry McDonald, chairman of the Reconstruction Finance Corp., Johnson said, "I hope the pattern will not be departed from in any future agreement entered into by this country."

He referred to the agreement signed between the United States and Belgian Congo tin producers to supply tin at \$1.20¾ per pound delivered to American ports. Bolivia is demanding \$1.35 per pound. Under the contract signed with Belgian Congo producers, a minimum of 7000 tons will be delivered annually. About 50 pct of production will now be aimed at the U. S.

A contract has also been signed with Indonesian producers. Similar offers are reported to have been made to Bolivia.

Calif. Tungsten Deposit Found with DMEA Support

Secretary of the Interior Oscar L. Chapman announced certification of the third significant tungsten discovery under the program of the Defense Minerals Exploration Administration.

The discovery was made on the holdings of G. McGuire Pierce in Dinkey Creek, Fresno County, Calif. The Interior Dept. said the deposit showed a considerable amount of commercially valuable tungsten. The discovery was made under the matching basis terms of the DMEA. The Government supplied 75 pct of the \$15,000 cost of the exploration project.

(Continued on page 450)

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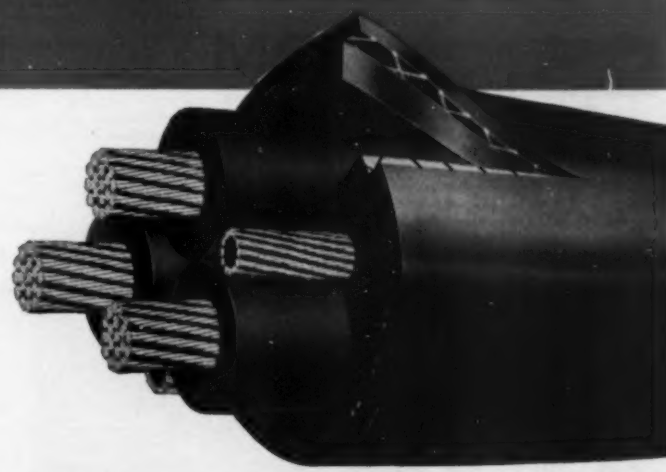

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United States Iron Ore Production Jumps 19 Pct to 105.2 Million Net Tons in 1951

United States iron ore mines produced an estimated 130.4 million net tons last year, an increase of 19 pct over 1950 production, according to the American Iron and Steel Institute.

About 105.2 million tons, or 80.7 pct, came from Lake Superior mines. Mines in that area accounted for 81.2 pct of total production in 1950. Western mine output rose 42.5 pct last year to 9.35 tons.

In addition to a greater increase in research aimed at higher utilization of taconite reserves, several iron mining projects of major importance are underway in Canada, Liberia, and Venezuela, and explorations are in progress in Peru.

The iron and steel industry expects to spend hundreds of millions of dollars for ore expansion within the next few years. Some investments have already shown results.

New properties in Venezuela shipped nearly 712,000 tons last year, while Liberian developments shipped more than 123,000 tons. The Cerro-Bolivar project in Venezuela and another in Quebec-Labrador are scheduled to start shipping to the United States in 1954.

In 1951 Chile led the field of ore exporters to the United States with 3.1 million tons. Sweden was second with 2.8 million; Canada, 2.2 million, and Brazil, 1.2 million. Venezuela was in fifth place, followed by Algeria, British West Africa, Mexico, Tunisia, and Liberia.

Plan Pilot Taconite Plant

Allis-Chalmers Manufacturing Co. is planning construction of a new $75,000 pilot plant for pellatizing and heat hardening taconite concentrates.

G. V. Woody, manager of the processing machine department, warned that a great deal of testing remains to be done before the commercial feasibility of continuous heat hardening of taconite pellets can be determined. The pilot plant will be a cooperative effort with the Arthur G. McKee Co. of Cleveland.

The plant is being built at Carrollville, Wis., 14 miles south of Milwaukee, and is expected to process one to two tons per hour of concentrated taconite fines. The fines will be brought in by rail from the Mesabi Range.

Pelletizing will be done in a balling drum. The green (soft) pellets will be predried in a rotating drum prior to being fed to the traveling grate. During the travel on the grate, the pellets will be further dried, preheated, ignited, and burned. Fuel consumption will be reduced by utilizing waste heat from the grate in the drying and preheating stages.

Study will also be made of heat hardening of magnetite and hematite concentrates without admixed carbon, Woody said.

It is planned to test the unit's product in blast furnaces in actual iron production.

Floods Lower Rhodesian Mine Production

Coal shortages compounded by floods are expected to cut copper production at the Mufulira mine in Northern Rhodesia during the current quarter, according to a company spokesman.

The coal shortage has been aggravated by flooding of the Kafue River, which threatens to damage the railway bridge crossing it. American Metal Co. of New York controls Mufulira through the Rhodesian Selection Trust. The bulk of the copper produced is sold to the British Ministry of Supply on the basis of a New York export price.

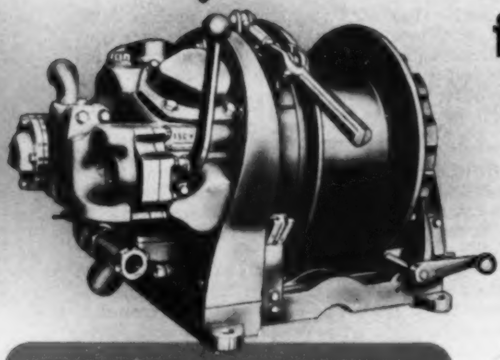
Ore Discoveries May Bring Venezuela Steel

The discovery of high quality iron ore deposits near Caracas, Venezuela, may lead to the development of a large scale steel industry near that city, said O. C. Laird, public relations officer of the Orinoco Mining Co.

Preliminary studies are now being made on the deposits. It was further reported that iron ore has been located near the town of Valencia in the State of Carabobo, also in Central Venezuela. Orinoco Mining is currently engaged in operations in the Cerro Bolivar "iron mountain" in Venezuelan Guiana.

SINGLE DRUM HOISTS

They're real Mining Hoists— designed and built from long field experience expressly for underground use



Joy Single Drum Mining Hoists are compactly designed, of modern, high-strength alloy metals, for lighter weight with greater power and efficiency.

The complete Joy line includes the "Turbinair" series which develop maximum horsepower per unit of weight; the "Pistonair" with reversible four or five cylinder motors; and compact, rugged electric-powered models which feature space-saving short-length external motors.

Joy Single Drum Hoists are built with lifting capacities ranging from 500 to 3500 lbs., and rope capacities from 150 to 1500 ft. "Pistonair" models have a positive acting safety brake which holds the load in any position when the throttle valve is "off."

Joy also manufactures a complete line of Shaft Hoists, and Two- and Three-Drum Slusher Hoists in capacities to handle all hoisting and scraping needs.

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Mining Engineering Trends

LATEST estimates of Russian steel production for 1951 give rise to the belief that on at least one vital front, the Western bloc of nations has maintained a definite advantage. On the surface, we are outproducing the Eastern bloc by more than twice. The United Nations Economic Commission for Europe estimates Russian output at 31.30 million tons, a 15 pct increase over the previous year. Production for the entire Iron Curtain is figured at 40.93 million tons. Western Europe alone produced 17 pct more steel during the same period. The figures are misleading in one sense. We have no precise idea of how much steel is being used for arms. The Western nations are still expending much of their production for consumer goods, and you can't throw washing machines in a shooting war. It is significant to note that Russian production is well beyond estimates made by world authorities prior to the current international situation.

INTERIOR Secretary Oscar L. Chapman warns that the United States must make use of the resources of other nations or face mineral impoverishment once the time of struggle is over. He pointed out that the present struggle between two ideologies can leave the United States in such a state as to preclude anything but political victory—with an accompanying economic loss that will be irreparable. Chapman scored the attitude whereby conservation programs are looked upon as a luxury, pointing out that the nation faces an ever increasing need for action on that front. He also announced that the Bureau of Mines doubled its facilities for producing pure zirconium during 1950-51 at the request of the Atomic Energy Commission. Zirconium may be an important structural metal for atomic power devices. The Interior Secretary also said that the AEC has provided the Bureau with funds for the study of radioactive minerals and to help test and evaluate about 9000 ore samples believed to contain uranium.

IT may be that original estimates of the Greater Butte project ore reserves were definitely on the conservative side. Current explorations indicate the area contains several million tons more than was first suspected. Millions of tons have already been added to the original estimate. Some sources expect a possible 100 million tons in addition to the starting figure of 150 million. If reserves to that extent are uncovered, it will probably result in another project operated concurrently with Greater Butte. Right now, the Kelley mine, a result of the Greater Butte project, is producing 2000 tons a day and figures to reach about 5000 tons by May. The Anaconda payroll at Butte has reached 7500 men per day, earning $2.8 million.

FIFTEEN hundred tons of slab zinc a year will be available for defense under the terms of a contract signed by the Defense Material Procurement Agency and Vernon C. Davis, Wisconsin mine owner. According to the contract Davis will construct concentrating facilities valued at $190,000, with a 250-ton per day ore capacity. The Government has agreed to purchase up to 3000 tons of slab zinc at 15½¢ per lb, providing it does not bring a higher price elsewhere in the nation.

COAL operators are getting a lower price for their product than in 1948 but costs are still climbing. One of the items especially tough to keep under control is the cost of conveyor belt systems. The belt conveyor is of considerable value in mining thin seams. A surface belt may last 15 years, but an underground belt has as short a life expectancy as four years. Rubber and canvas costs have been flying on jet propelled wings in recent years and replacement cost has gone along for the ride.

One answer may be longer belt conveyor centers, which tend to eliminate transfer points resulting from a series of short conveyors. It means less machinery, reduced degradation of coal, and less dust. Neoprene steel reinforced belts, another development, may lead to stronger belting at lower prices. The material, with some of the characteristics of rubber, is more fire resistant, able to stand 450°F before charring.

Germany has tried stainless steel endless belts, with indications that they may last longer than conventional rubber. A British development is the cable belt conveyor. It uses two wire cables to carry supporting cross connectors acting as a road bed for a light, inexpensive belt.

CHARLES E. WILSON'S last report to the President on progress of defense mobilization contains one small item which deserves long consideration by the public, and those directly concerned with building the muscles of democracy. Wilson says that increased availability of materials will have no effect on the armament program speed-up. From the beginning the military has received all the materials required by their schedules. Invention and design remain the limiting factors. Constant changing of plans in no small measure has led to a road block for which industry has, in some quarters at least, been given the blame.

In general, however, the report contains a strong undertone of confidence in the nation's ability to make the grade. Foreign aid in several forms seems to be on the increase. Wilson notes that Venezuelan iron ore production in 1952 is expected to reach one million tons, double that of 1951. The report estimates we will be getting about 17 pct of our iron ore supply from foreign sources. The report, in line with the expected increase of alien suppliers, restates the case for the St. Lawrence Seaway. It cites the need for a water route from the Quebec-Labrador area to points in the northern part of the United States.

The report also announces that the aluminum industry, with the aid of government, is scheduled to add another 415,000 tons of primary capacity. To meet the general increase in transportation demands brought about by increased production in all metalworking and mining pursuits, an increase in the manufacture of all forms of rolling stock is one of the pressing demands of the near future.

BUREAU OF MINES METALLURGISTS at Rapid City, S. D., are trying to figure out a way of using some 12,000 tons of low grade beryl lying in Black Hills, S. D. The metal has a wide variety of direct uses, and is also important as an alloy. But the beryl found in South Dakota has until now defied economic recovery. It was first reported in 1914, but has been produced spasmodically since then. Only 104 tons were mined in 1949.

WESTERN Germany set a postwar record for coal production last February, with an average of 408,000 tons per day. New mines have either been brought into operation or have increased their tonnage per day since then. With the growing production rate in Great Britain, the U. S. Mutual Security Agency is scheduled to have some of the weight removed from its broad back. The agency has supplied about 25 pct of the Western European dollar purchases of coal for essential industries. A drop to less than 2 million tons a month of bituminous is expected from the record 3½ million tons shipped from the United States to Europe in October 1951 and January 1952. The effect the decrease will have on the local market cannot be estimated in the immediate future, and thus recent contracts have been on the conservative side.

Certainly, it will have a good effect on Western Europe's dollar balance, releasing currency for other needs which until now have had to play second fiddle to more vital necessities. Total European needs for 1952 are estimated at 25 million tons at this time, compared with previous figures of 30 to 40 million tons. Weather conditions and the state of world politics, can of course, alter the latest estimate drastically. In the meantime, government controls on coal exports, aimed at relieving congestion at Hampton Roads, have been removed.

IMPROVEMENT in the scrap metal supply has been spotty, with some areas reporting definite signs of recovery from the winter months, while others still languish in a state of uncertainty. Spring was expected to give collections the needed shot in the arm, but while some mills report 45-day inventories, others must promote scavenger hunts to keep stockpiles within some reasonable state of health. Contaminated scrap has also been reported in some quarters and has been a factor in slowing up development of a margin of safety for mills. The situation in general, however, is decidedly better than during winter months, and can be expected to improve considerably as the weather gets warmer.

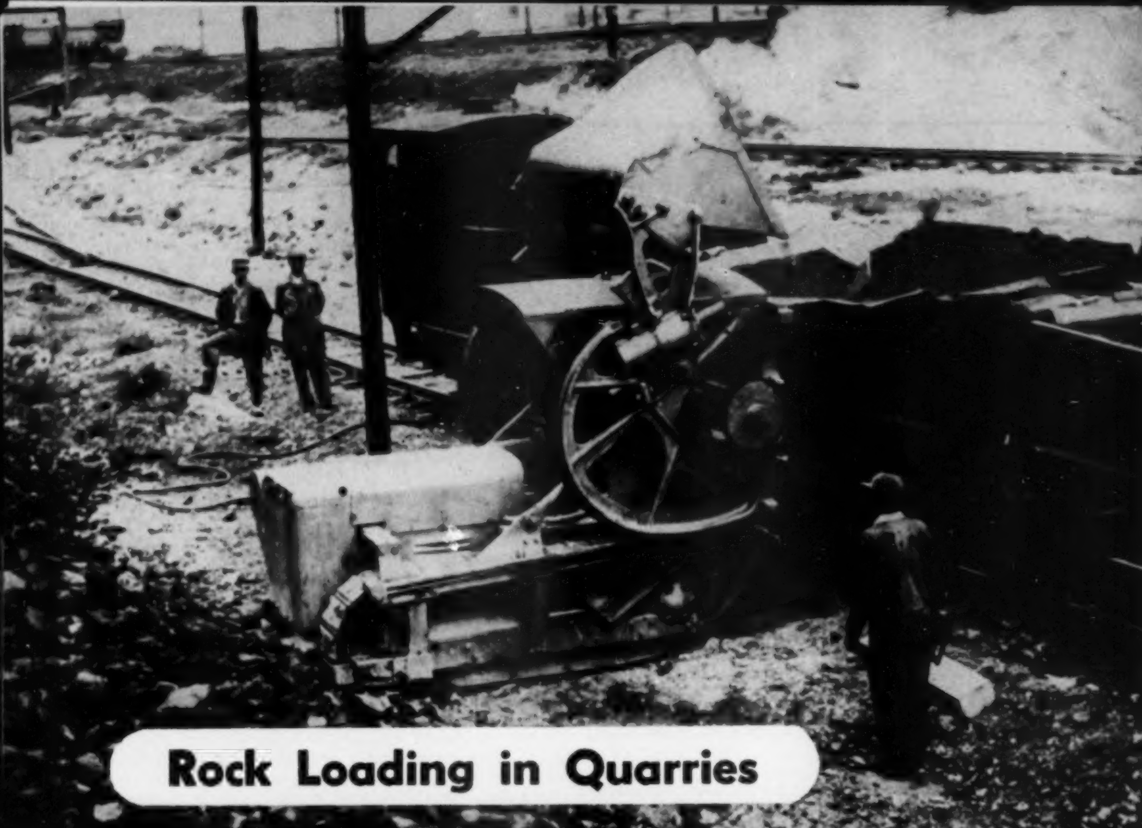
FATHER JAMES B. MACELWANE, St. Louis University geophysicist, has discovered that the earth gets an occasional case of D.T.'s for some unexplainable reason. The minute tremblings in the earth's skin were detected on the University's seismograph at Florisant, Mo., but despite the possibility of relation to the weather, they are different from the moving tremors associated with hurricanes and other ocean disturbances. Because of the prohibitive cost of monitoring the phenomena, each 24-hr period is being sampled. The disturbances rise and fall in amplitude for no reason yet discovered. The same records have been produced over a wide area of the earth's surface. Father Macelwane also reports study of the different type micro-oscillations caused by ocean storms has given rise to the belief of possible submarine barriers which block penetration of the waves. How the storms transmit the waves still remains a mystery.

NICARO, the United States Government-owned nickel producing plant in Cuba, comes in for a big play in the April issue of the magazine *Fortune*. After reading the article, written by Herbert Solow, it wouldn't be impossible to believe that no nickel is forthcoming, nor is there any immediate prospect of production. The truth of the matter is that Nicaro, now operated by the Nickel Processing Co., has been in production since Jan. 31, 1952. Full production is expected by June 1, with an annual output of 15,000 tons. The *Fortune* article goes into great detail concerning the award of the contract. In the process, it manages to bury the fact that the plant is operating and is producing ahead of schedule. Billiton Maatschappijj, the Netherlands tin company, owns 50 pct of Nickel Processing, while National Lead Co., has 30 pct, and a Cuban group owns the remaining 20 pct.

BOHN Aluminum & Brass Corp., of Detroit and the Glidden Co., of Cleveland have joined forces in a broad titanium research program. An agreement signed by R. J. Roshirt, executive vice-president of Bohn Aluminum and J. P. Ruth of Glidden, provides for pooling of the research facilities of both organizations on a 50-50 basis. Glidden and Bohn have worked independently on titanium research for some time and are past preliminary stages. First objective of the program is to produce pure ductile, titanium. A number of methods are presently known for the production of titanium, but all of them are prohibitive in cost.

The Glidden Co. owns ilmenite-bearing lands in North Carolina and has operated one of the largest ilmenite mines in the United States near Lenoir, N. C. for the past 10 years. A 100 million lb annual market is foreseen by experts. The North American continent is sufficiently wealthy in rutile and ilmenite ore to guarantee self sufficiency for many years.

The importance of titanium has been compounded in the last few years with the trend of the air world toward jet propulsion. It is also used for submarine construction and has almost unlimited possibilities in the building trade. It may be the metal which will actually make air freight a practical and common part of our system of transportation. Aircraft engineers have estimated that it will increase air payloads by about 400 pct.



Pictures show a typical European installation where Eimco Loaders are contributing to production for a free world.

EDITORIAL

THE CROSS ROADS

COLLECTIVE bargaining, heretofore loudly proclaimed as one of the stout timbers of the Republic, has passed from the picture. The *coup de grâce* was struck by the President of the United States when he ignored his vested legal prerogative and seized the steel industry by the dubious authority of his "inherent powers." Autocratic judgment by a single official replaced arbitration by the steel industry and labor union. This is the outcome of a battle which has been bitterly waged for months in the hazy atmosphere created by well-oiled propaganda machines.

Prior to government intervention, there was no clear picture of the issues at stake. All parties concerned marshalled their accountants to conjure up figures proving their respective contentions. The results were fed into public relations departments for embellishment and dissemination. Thus, the public was asked to believe on one hand, that the steel worker, who is second only to the coal miner in hourly wages, was entitled to an increase in pay and other benefits. Nothing was said about the effect of such an increase on the national economy. Industry in turn pointed to declining profits after taxes, but failed to mention the more than a billion dollars spent on plant expansion, in addition to the payment of substantial dividends for 1951.

With the intervention of the administration, the immediate issue became clear to freedom-loving Americans. Is the executive branch of the government above the law, or is it supposed to uphold and carry out the laws of Congress? The present administration forgot that it is a servant of the public and has become despotic. The American people must obliterate this dangerous precedent which has been set by the administration or accept the ominous consequences of submission. Fortunately, the strongest weapon of the citizen—the ballot—still remains.

When democratic procedure is restored, stabilization of the economy will continue to challenge us. This is the main public stake in the steel controversy. Only a solution based on bed-rock democracy will work. If the emergency is real, and we must assume that it is, this is the time for sacrifice. Industry and labor must meet without rancor to resolve wage contracts through collective bargaining. The government role, under emergency conditions, must be to objectively maintain wage and price ceilings in the interest of stabilization. Political expediency cannot be permitted to influence these controls. Politics and not the public weal have been the guide lines for too long.

We are at the cross roads. By all sober judgment, this is the time for Americans to take a militant interest in *their* government today. Tomorrow will be too late.

The hills of Santa Eulalia in Northern Mexico, where the Conquistadores once brought fire and death because of their lust for gold. There was no gold in Santa Eulalia, but centuries later, the San Antonio mine is producing a variety of metals more significant today than the yellow sand sought by the Spanish. In the foreground can be seen the mine. The Chihuahua smelter is in the background.

San Antonio Mine — Landmark On the Path of the Conquistadores

by C. M. Signer and W. P. Hewitt

THIS is a story of a mine discovered in the days of the Conquistadores but that remained unimportant until the second decade of this century. Without the usual legendary history of romance and fabulous riches it has become famous for the development of unexpected metals and unusual problems. These developments and experiences, exciting and interesting as any legend, are the substance of the San Antonio mine of the Santa Eulalia district in northern Mexico.

Originally opened for its silver-lead content, it developed a tin deposit of singular economic importance and unique geologic interest, a vanadium orebody that for a few years placed it among the leading vanadium producers of the world, and now lead and zinc sulphide ores. It has been the scene of a disastrous flood that brought deep-sea divers into Mexico's desert mountains and raised important problems regarding ground water, demanding unusual measures.

MR. SIGNER and MR. HEWITT are Superintendent and Geologist, respectively, of the San Antonio Mine, American Smelting & Refining Co., Santa Eulalia, Chih., Mex.

Geology and operating questions relating to the above matters will be discussed and much of it is recent history which has become a part of Mexican mining lore.

History Dates Back to 1325

Santa Eulalia is the oldest of American cities, founded in 1325 by the Aztecs under the name of Tenochtitlan and devastated in 1521 by Cortes and his rapacious band of iron-willed adventurers. Their lust for gold burned through the Aztec empire and onward into most of North America. Santa Eulalia was in the path of conquest. In 1591 the district's first ore discovery was made near what is now the San Antonio mine. The ores carried no gold and the silver-lead content was unattractive to the early pioneers. Not until 1700 was real mining activity undertaken.

Santa Eulalia, in the central part of the State of Chihuahua, and some 20 miles east of Chihuahua City, lies in a rugged limestone range that rises 2200 ft above surrounding valleys. Of the flanking valleys, that on the east is long and wide, flat floored; and choked with unknown depths of sands and gravels. It slopes gently to the southeast. In it are numerous wells furnishing irrigation water for a large acreage. The western valley is smaller, in large part floored with volcanics, and test wells to over 800 ft have found no water bearing strata. Its main drainage feature is the Chuviscar river which flows northeast into the eastern valley and then turns abruptly to the southeast for its confluence with the Conchos river, approximately 25 miles southeast of the San Antonio Mine.

The climate is semi-arid with an average rain fall of 14 in. Vegetation is grass, thorny brush and cactus, with a few scrub oaks along the summits. There is little soil and no permanent surface water.

The Santa Eulalia range is a series of gently disturbed lower Cretaceous limestones, approaching 3000 ft in thickness. Overlying the limestones is a thin mantle of limestone conglomerate and tuffs, probably of Tertiary age. Near the bottom of the limestone series, and interbedded with them, are numerous sills, both rhyolitic and andesitic. The ore deposits which are replacement bodies in limestone, have produced many millions of tons since the 1700's, and have placed the area among the great lead districts of the world.

Tin Discovered

The San Antonio mine, at an elevation of 5200 ft above sea level, lies on the eastern edge of the district in a small north-south valley which is the physiographic expression of a graben. Cutting the graben is a series of northerly trending rhyolitic dykes, collectively known as the San Antonio dike, and the mine is associated with them. The ores have come predominantly from large chimneys and numerous small bodies banked along intrusives. In the upper levels, replacement veins isolated from the dike contacts, supplied important tonnages. With the exception of the replacement veins all ore bodies are a varying mixture of garnets, epidotes, fibrous silicates, and ore. This is in contrast with all other mines in the Santa Eulalia district, which have clean lead carbonates in the upper horizons and lead-zinc-iron sulphides at depth, usually uncontaminated with other minerals.

Originally operated for its lead carbonates which were considered exceptionally clean and very desirable for direct smelting, it provoked consternation in the '20's when tin was discovered as an impurity in the lead bullion received at the refinery. This was rapidly traced to the Chihuahua smelter which then tested all receipts until finally, in her own front yard, she discovered a high grade cassiterite content in the least suspected of all ores, those of the San Antonio mine. This brought about the construction of a concentrator, and revolutionized a sleepy mine into the one and only formal underground tin producer on the North American continent.

Although San Antonio became an important source of tin, tin minerals were rarely recognized in hand specimens. While her stopes averaged 1.5 to 2.0 pct tin, a man could work for days in stopes as high as 8 pct and never see an indication of a tin mineral. Yet, tin ores occurred with every type of mineral association, but not with any particular mineral or type of ore.

Milling operations showed that the tin mineral was a fine grained crystalline cassiterite, brownish yellow to slightly purplish in color; from petrographic studies that it occurred microscopically as free individual crystals, also as intimate intergrowths associated with hematite, quartz and topaz, and to a minor extent with magnetite, ilmenite, columbite, muscovite, tourmaline, and calcite.

There were two main sources of production that carried tin from top to bottom; the Tin chimney and a series of replacement veins known as the Dolores fissures. Elsewhere it was limited to definite horizons, as in the Cocks ore body which in its upper part had a tin content comparable to either the tin chimney or the Dolores fissures, but which in its lower horizon lacked tin. In addition other small bodies produced ores of good grade.

The tin chimney had an average diameter of approximately 130 ft, was over 1000 ft deep and its top lay within 200 ft of the present erosion surface. Its upper 130 ft were within conglomerates, its remaining depth in limestones, and it passed from one to the other without change in character of ore. Throughout most of its length it lay against the east wall of a rhyolite dike, but in depth this disappeared while the tin chimney in greatly reduced cross section wandered off to the southeast. Beneath the main body of the chimney, other rhyolite dikes appear. Associated with them is a large mass of lime-iron silicates containing patches and disseminated grains of sphalerite with slight galena and pyrite. These silicates, unlike the ores of the chimney proper which were heterogeneous mixtures of iron oxides, lead carbonates, and secondary zinc, are low in tin. Within the chimney, silicates occur, but only in erratic and minor concentration.

In the upper 350 ft of the chimney a vein breaks out of its southeast wall, strikes southwest and rises flatly toward the east. After a length of roughly 350 ft it begins splitting into a series of veins known collectively as the Dolores fissures, which diverge to the south and lie within a fan-like belt 1000 ft long by 350 ft wide. They are mineralized faults of small throw, cutting the capping series and extending downward into the underlying limestones where they are lost. Ores are the typical mixtures of lead carbonates and iron oxides, but they also carry a tin content of the same general grade as those of the tin chimney. They contain no suggestion of lime-iron silicates.

The third major source of tin was the Cocks ore body, a chimney over 1000 ft deep and 100 ft diam. Its oxide ores varied from a silver-lead-tin zone at the top; through a lead-vanadium zone in the center; to a silver-lead horizon at the bottom. The silver-lead-tin zone was essentially silicate-free, and its ores were comparable to those of the tin chimney. The two lower zones carried little tin, and abundant silicate minerals at the periphery of the ore. Mixed with these oxides were numerous residual boulders of galena.

It was in the Cocks orebody that vanadium ores were first discovered, but once vanadium consciousness awakened ore was found in many parts of the mine. All discoveries were confined to a definite 200-ft zone approximately 1000 ft below the surface. These ores were the product of oxidation and migration that had been deposited in pre-existing voids. They varied from yellowish coatings, through yel-

lowish fine grained sandy beds, to manganese muds interstratified with vanadium bearing streaks. From assays we know they were vanadinite, descloizite, and mimetite. Their origin is not certain but a possible correlation exists between overlying oxidized silicates and underlying vanadium ores.

Both the Cocks orebody and the tin chimney, as well as numerous others not described, passed into sulphides some 1300 ft beneath the surface. There are several sulphide types, all of them mixtures of lead, zinc, and iron. There are massive mixtures of galena, marmatite, pyrite, and slight chalcopyrite that represent direct extensions of the upper level oxide bodies. There are also important sulphide bodies disseminated in a silicate gangue. These are predominantly a brownish sphalerite of variable concentration, with minor amounts of galena, pyrite, and chalcopyrite. Most San Antonio sulphides are tin free although the extensions of the tin chimney carry a low, variable cassiterite content. Data that would pin down the genetic history of the tin ores is lacking, but it is felt they are related to the end stage of the sulphides, and not to the silicate minerals.

Cassiterite Concentration

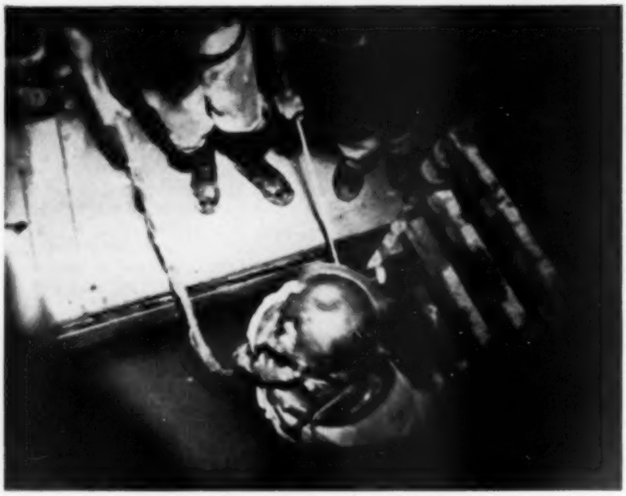
With the discovery of tin in the San Antonio ores, research work was commenced on a process for the recovery of the lead carbonates by flotation and of the cassiterite by gravity and magnetic concentration methods. This work occupied the period 1924 to 1927. Long hours were spent in laboratory experimental work and every reagent known to the science at that time was tried, but with little success. Soluble salts and slime interfered to give a low grade lead concentrate, carrying a large percentage of gangue, and of course tin. However, it was discovered that the orthodox practice of conditioning with sodium silicate or sodium carbonate before the addition of the sulphidizing agent was detrimental, and in keeping with its tradition of being different, the San Antonio ore required the reversal of the established order. Instead it required sulphidizing first, and conditioning with sodium silicate after thorough sulphidization. This simple discovery changed it from a most difficult ore to one of the most easily floated ores of its time. With a minimum of control we obtained a better than 90 pct recovery in a concentrate assaying approximately 62 pct lead. Table concentration of the lead float tailing produced a concentrate of cassiterite and hematite, assaying approximately 20 pct tin. This product was then dried and treated in Wetherill type magnetic separators to produce a tin concentrate assaying a little over 60 pct tin.

A 200-ton per day mill for the treatment of the ore along the above lines was built and placed in operation in 1930. After a few years of routine operation on ores similar to those originally tested, the zinc content of the ores increased to a troublesome degree. The zinc minerals concentrated out with the cassiterite in both the gravity and magnetic sections to seriously contaminate the tin product. This was remedied by the installation of a sulphuric acid leaching section which gave a final cleaning to the tin concentrate.

In 1936, when the method used in the treatment of the lead-tin ore was well established, another problem arose. A bright yellow mineral in large quantity suddenly appeared in the tin concentrate streak on the concentrating tables. This proved to be lead vanadate. It would not float with regular reagents and passed on through the flotation circuit to the tables, contaminating and completely ruining the tin concentrate. It was traced back to an area in the mine that had been producing clean ores for many years, but was now destined to produce many tons of a new type of ore—a lead sulphide, lead carbonate, vanadinite ore. After months of research, a flotation scheme was developed for the separation and recovery of the lead sulphide and carbonate minerals as one product, and of the vanadinite as another. Fortunately the ore had no tin to make it more complex, and for the next few years the mill handled both lead-tin and lead vanadium ores in alternate periods.

By 1940, after the production of 95,869 tons of lead carbonate, 8752 tons of vanadium, and 4423 tons of tin concentrates, both the lead-tin and the vanadium ores were completely exhausted and operations suspended. The development of the few occurrences of sulphide ores was not warranted until several months later when they were made attractive by the war. Exploration work on the ninth and tenth levels beneath worked-out oxides developed bodies of high grade zinc sulphide ores, associated with the metamorphic garnets and other silicates along the dike contacts. In the meantime the old tin mill, had been remodeled for the purpose of treating 600 tons per day of lead-zinc sulphide ore from the

Looking south along the Santa Eulalia range, the San Antonio colony can be seen in the left center and the gravel choked eastern valley in the upper left corner. Graben extends from bottom center to upper right.



A diver, hundreds of miles from the nearest sea, starts down a ladder into the flooded lower levels of the San Antonio mine. Divers placed a concrete bulkhead 40 ft back from the punctured ninth level face, in order to seal off the water so that the mine could be worked.

western section of the Santa Eulalia district. In 1942, a 200-ton section was added for the treatment of the San Antonio sulphide.

The conventional routine of mining sulphides flowed smoothly until the San Antonio mine again flaunted her unpredictable character—this time with dark tragedy. On the night of Nov. 10, 1945, after the blasting of a dry face at the south end of the ninth level, some 1450 ft beneath the surface, water with the volume and horror of a tidal wave engulfed the lower levels, and in the short period of 2½ hr rose to within six ft of the eighth level. Thirty men were trapped by the surging waters. Miraculously, 24 escaped, some by climbing ladders and others by swimming up stopes with the rising waters. The remaining six were drowned.

San Antonio had always been considered a dry mine and there was no reason to suspect hidden waters. It had produced only 150 to 200 gpm from various scattered sources, the volume fluctuating with the rains. Furthermore, exploratory diamond drilling ahead of the advancing face had given no indication of water.

Since transformer capacity was sufficient for the handling of 1300 gpm, pumping equipment with this capacity was installed. After three months of pumping at this rate the water level was lowered a mere 22 in. The volume was enormously greater than the original estimate, and only by sealing it off could the mine be recovered. This was accomplished by placing a concrete bulkhead 40 ft back from the punctured ninth level face. Its construction involved the sinking of a winze from the eighth to the ninth level, and the use of this winze for the pouring and depositing of concrete under water through vertical or steeply inclined pipe-lines. Deep-sea divers were engaged for the operation.

Ground conditions required the sinking of an inclined instead of a vertical winze, and its sinking presented no particular problem until the drilling and blasting of the connecting round. This involved the complete destruction in a single blast of the 7 ft rib of rock that remained between the bottom of the winze and the water filled ninth level. Several times workers were driven out by water welling up through fractures, but finally the purpose was achieved by drilling a 59 hole round through 2-in. pipe nipples cemented into a concrete pad which was poured over the bottom of the winze. This arrangement was necessary to seal off the inflowing waters and to facilitate the control of water that might be struck in any of the holes. It also insured the proper spacing alignment of the holes, each of which had to reach a designated point within the block. When the round was blasted, water immediately filled the winze, but how completely the rib had broken was not learned until the divers cleared out the broken rock and descended freely into the drift.

The divers, on inspecting the winze, found there had been sloughing of the walls, which had completely filled the lower 30 ft. They were faced with a full scale mucking operation. It had to be done under many feet of water, in total darkness, and handicapped with a bulky, cumbersome diving suit. Some of the muck was removed with a 6-in. air lift, the diver working it into the intake with a high pressure water jet, but much of it had to be shovelled and hoisted in a small bucket.

In the course of six weeks they reached the ninth level, 83 ft beneath water level, and reported the connection had been made successfully. It was on line to the full size of the winze, but there was considerable overbreakage along the sides necessitating the removal of a greater quantity of broken rock and the construction of a larger bulkhead than anticipated. The divers jetted the muck along the drift, leaving a thoroughly cleaned 10-ft section along the floor, and built sandbag walls between the floor and the back to retain the concrete to be poured. They groped their way to the punctured heading and found the water had not entered through the face but had come from a hole 2 ft wide by 4 ft long, in the right hand side of the drift some three ft back of the face. Whatever had been tapped, the preceding round had missed it by a bare 6 in.

The pouring was done through an 8-in. pipe extending from the top of the winze to the floor of the drift below. This pipe line had to be maintained

full of the proper mixture of concrete, allowing it to ooze slowly out the bottom to form a cone around the end. Eventually the pressure on the exit would be equalized and the flow would stop. It would be necessary to raise the pipe a foot or so, taking every precaution to assure that the discharge end remained embedded in the growing cone of concrete, and that the incoming material was injected into the center of the pile. Otherwise the column in the pipe would discharge freely and become ruinously diluted with the water.

In order to keep the pipeline empty and watertight before being filled with concrete, the bottom was sealed with a tightly fitting disc, slipped off quickly by the diver at the start of the pour. The upper section of the pipeline consisted of two-ft nipples, flanged together to allow for easy removal. To insure a full and steady flow, the top was provided with a receiving hopper of ½-yd capacity, and the line was manipulated by means of a small hoist and chain-blocks. As success depended largely on the rapidity of pouring, the concrete was mixed in two 5-ft motor-driven mixers set up in tandem at the collar of the winze. Materials were mixed dry in the first machine and wet in the second. A −1-in. aggregate was used in a 1-3-2-½ mixture with a small amount of *Pozzolith*. After a short rehearsal the actual pour, consisting of 59 cu yd in 305 batches, was made in 8 hr. This filled the walled off section of the drift and reached up into the winze to a point 25 ft above the drift floor. To insure that all corners and irregularities along the walls were properly filled, the diver inspected the filling at regular intervals. The subsequent unwatering of the mine proved that the dam was well bonded to the back, walls, and floor, effectively sealing off the flood waters. Its leakage, mostly from the floor contact, amounted to only 80 gpm, but within three years this increased to 300 gal. For additional precaution a water door was placed in the drift 100 ft ahead of the bulkhead, and in anticipation of some future attempt to drain the sealed area, suitable nipples and valves were provided.

After the unwatering, considerable time was required in the removal of silt and mud that blanketed all drifts and workings to a depth of several feet in some places. Before mining operations were resumed, precautionary measures against any similar disaster were devised. The entire mine below water level was divided into individual compartments, separated by water doors and concrete bulkheads. A routine was established whereby all blasting is done behind closed doors under the supervision of a blasting boss. In this way, if a water course is encountered in any blast the flooding will be confined to one section of the mine, and with little possibility of loss of life. Also, all headings are preceded by long diamond drill holes to test for water courses ahead of the face. With these precautions, since the resumption of operations in October 1947, ore breaking and development operations have been carried on without incident or interruption. However, the fear engendered by close association with a treacherous and unpredictable body of water has lead us to attempt the unwatering of the ground behind the dam.

Early in 1951, we obtained the power and installed the equipment for pumping 3000 gpm. The pumps were placed at the ninth level station. To take advantage of the hydrostatic head they are connected directly to the valves of the waterdoor in front of the dam by means of a 16-in. intake line 1340 ft long. The discharge, also 16-in. diam, extends 1065 ft up the shaft to the 0 level and then 900 ft along this level to discharge by gravity into an arroyo on the north side of the mine. The tapping of the flood waters involved the blasting and destruction of the bulkhead. An additional water door was installed in the drift a few feet ahead of the main door. To serve as a relief vent in the section between the doors and the dam, a connection was driven from the eighth level. The blasting of the bulkhead was done by means of diamond drill holes from the eighth level, not however, without several unsuccessful attempts. The resistance of the bulkhead, which had caused so many diffculties during its construction four years previously, was full testimony to the perfection of the divers' work.

Since March, when the pumps were started, the water level has been lowering 3 to 4 in. a week. A rather slow rate, but it's progress and it is hoped that after a reasonable period of sustained pumping the rate will accelerate. As in the days of the initial pumping attempt, the origin and magnitude of the water is still unknown. Possibilities include a vast series of caves, the normal water level of the entire surrounding country, a perched water table, and a hot spring, partially of magmatic origin.

From the rapid inflow of storm waters it can be assumed that a direct connection to the surface must exist. In 1946, less than 24 hr after a 1½ in. storm the water level rose 6 in. and the divers reported extreme murkiness. During September and the first few days of October, after 10 in. of rain the water level rose 4 ft 5 in. to a point higher than the flood of the previous November. In addition to this rapid influx there is a more gentle rise that continues for several months after the rains. We also know that the water level lowers slowly during the dry season.

The surface contour corresponding to the flood level lies far out in the valley to the east of the mine. East and southeast of this contour the valley floor drops 300 ft through an area 15 miles wide by 30 miles long and reaches the Conchos river. Records of valley wells indicate discontinuous aquafiers, not a uniformly persistant water table. These facts suggest the flood waters are from a perched table, independent of the valley waters. Undeniably there are caves because as a result of diamond drilling, since disaster, one was found directly adjacent to the punctured face.

In the 2½-month period of pumping before the sealing of the flood waters there were periodic cycles of irregular fluctuations of the water level. It was not determined whether these were due to vacuum effects, or had a more subtle origin.

The temperature of the flood waters was 93°F and since hot springs exist in the valley east of the mine, a possible correlation exists. However a comparison of spectrographic analyses between San Antonio and reginal waters, while suggesting the possibility, were largely inconclusive. The temperature could be due to underlying recent intrusives.

In time the extent and the origin of the water will be found, and the biggest of San Antonio's many and varied problems will be solved. These, in their time, have furnished their share of worries and distress, but in retrospect they have proven to be experiences of great interest, and their solving has contributed to the techniques of mining and to the romance of Mexican mining lore. Without doubt San Antonio's aura of unpredictableness still exists and will continue to challenge the imagination with problems.

Real del Monte finds:

Low Base Metal-High Silver Ores Give Better Smelter Returns With Pre-cyanidation Treatment

by R. R. Bryan

SINCE the first applications of cyanidation to silver ores about 1906, treatment of ores in the Pachuca district has been entirely by straight cyanidation. Until about the year 1921, Real del Monte removed a sulphide concentrate (largely pyrite) by tabling in the cyanidation grinding circuit.

The ores milled prior to the year 1935 were dry ores, ideal for cyanidation, and yielded 90 pct of the silver and 92 pct of the gold.

Since 1934 there has been a notable increase in the refractivity to cyanidation of the average ores received by the mill. Tonnages of the former amenable ores have gradually been replaced by tonnages of more refractory ones, so that present average recoveries have fallen to 83 pct of the silver and 90 pct of the gold.

During this period there has also been an increase in the content of base metal sulphides in many of the ores although this increase is not always directly related to the increased refractivity.

Since March 1950, the Cia. de Real del Monte has been developing a flotation treatment for some of its ores before cyanidation. Suitability for flotation is judged not only by the base metal content, but also by the amount of additional recovery of silver and gold made possible by the combined treatment.

Selection and segregation of ores most suitable for flotation before cyanidation is a difficult problem for the mines, and the best that can be done is a compromise that leaves much to be desired.

MR. BRYAN is Superintendent of mills of Cia. de Real del Monte y Pachuca, Pachuca Hgo., Mex.

The Development of Pre-cyanidation Flotation at Real del Monte

Much laboratory work has been done on flotation of ores from the Pachuca district, however, the data could not supply reliable information on the many factors connected with the economics of low base metal-high silver ores. Pilot plant operations would be required.

Pilot plant facilities were made available in March 1950, when it became necessary to close down Loreto's 100 ton per day selective flotation plant because of exhausted ore supply.

The results of the 100-ton pilot plant tests indicated that a large part of our tonnage of cyanidation ores could be profitably treated by flotation, and a tonnage of 1100 tons per day was selected as the maximum tonnage of ore that the mines were likely to deliver, and because this tonnage fitted with the grinding capacity of this circuit.

Equipment was ordered for the production of a single flotation concentrate. The conception of flotation before cyanidation, at that time, was the production of a single concentrate to be shipped to the lead smelter.

The 100-ton pilot plant was immediately increased to 350 tons by installing additional flotation cells. This plant was operated continuously until the 1100-ton plant was put in operation in January 1951.

In July 1950, an increase in the quotation for zinc called for an investigation of the possibilities of producing a zinc concentrate, and laboratory tests on the tailings from the 350-ton plant indicated that a

General view of upper part of Loreto Mill showing San Juan head-frame, ore bins, crushing and grinding plants. In center and right foreground are the flotation, filtration and part of agitation units.

good zinc concentrate could be produced. This data only applied to the 350 tons being treated in the flotation plant. The remaining 750 tons of ore that would be supplied by the mines to make the 1100 tons for the large plant might be of considerably different character and assay. This question could be answered only by actual operation, and such operations were not expected to begin until January 1951.

Equipment for the production of a zinc concentrate were rougher and cleaner flotation cells, thickener and filter. By again borrowing equipment from the old 100-ton selective flotation plant it was possible to produce a zinc concentrate by purchasing only zinc rougher cells. If the project proved profitable, the cleaner cells, thickener and filter could be secured later and the 100-ton selective flotation plant free for operation with other ores.

This program was decided upon and the equipment was installed and operating by February 1st, 1951. Operation of the zinc section of the 1100-ton plant during February and March proved that a zinc concentrate could be made profitably. This operation period showed that greater tonnages of zinc concentrate could be made if the zinc concentrate were cyanided before shipping to the smelter.

Metallurgical Results of Pre-cyanidation Flotation

Metallurgical results are presented for the two cases in Table I.

Case I. Actual metallurgical results for the month of July 1951, without cyanidation of zinc concentrate.

Case II. Actual metallurgical results from short mill runs using the flotation flowsheet with cyanidation of the zinc concentrate.

Both cases are based upon the heads assays of July 1951.

Table I. Metallurgical Results of Pre-cyanidation Flotation

Case I
Mill Metallurgy for July 1951

Products	Pct Weight	Assays: Ag (G)	Au (G)	Pb (Pct)	Cu (Pct)	Zn (Pct)	Fe (Pct)
Pb conc	0.687	26,644	143.8	32.5	3.74	21.0	7.8
Zn conc	0.328	5,808	25.0	4.46	0.93	52.62	4.9
Tails	98.985	93.2	0.515	tr.	tr.	0.72	—
Heads	100.000	294.30	1.58	0.24	0.029	1.03	

Products	Distribution: Ag	Au	Pb	Cu	Zn
Pb conc	62.19	62.54	93.00	88.62	14.02
Zn conc	6.47	5.19	6.08	10.69	16.76
Tails	31.34	32.27	0.92	0.69	69.22

Case II
Mill Metallurgy—cyanidation plant for zinc concentrates, based on actual mill results for periods of several days.

Products	Pct Weight	Assays: Ag (G)	Au (G)	Pb (Pct)	Cu (Pct)	Zn (Pct)	Fe (Pct)
Pb conc	0.471	31,089	167.9	45.0	5.00	9.4	8.0
Zn conc	0.544	10,232	51.4	4.75	0.95	50.11	5.62
Tails	98.985	93.2	0.515	tr.	tr.	0.72	—
Heads	100.000	294.30	1.58	0.24	0.029	1.03	

Products	Distribution: Ag	Au	Pb	Cu	Zn
Pb conc	49.75	50.04	88.33	81.38	4.30
Zn conc	18.91	17.69	10.75	17.93	26.48
Tails	31.34	32.27	0.92	0.69	69.22

Case I

Pre-cyanidation flotation

Net smelter returns

Quotations: Ag, 90.16c per oz; Pb, 21.04c per lb; Cu, 27.3c per lb; Zn, 17.5c per lb.

Lead concentrate: 0.687 pct wt; 26,644 g Ag; 143.8 g Au; 32.5 pct Pb; 3.74 pct Cu.

Per Ton Concentrate

	Payments		Deductions			Net	Net Per Ton Ore
Ag	$733.71		Taxes:	$229.65		$504.06	$3.403
Au	153.58		Taxes:	30.56		123.02	0.845
Pb	126.22	= $142.37	Taxes:	37.02	= $47.83	94.54	0.649
Cu	16.15		Treatment:	5.81			
			Freight:	5.00			
			Total net smelter returns, pb conc			$721.62	$4.957

Zinc concentrate: 0.328 pct wt; 5808 g Ag; 25.0 g Au; 52.6

Per Ton Concentrate

	Payments	Deductions			Net	Net Per Ton Ore
Ag	$117.85	Taxes:	$37.12		$ 80.73	$0.265
Au	19.64	Taxes:	3.92		15.72	0.051
Pb&Cu		Taxes:	6.00			
Zn	169.41	Taxes:	27.93			
		Treatment:	54.01	= $108.49	60.92	0.200
		Freight:	17.06			
		U. S. Duty:	3.48			
		Total net smelter returns, zn conc			$157.37	$0.516

Net bullion returns from cyanidation of zinc conc.

None
Zinc conc not cyanided in this case

Case II

Pre-cyanidation flotation

Net smelter returns

Quotations: Ag 90.16c per oz; Pb, 21.04c per lb; Cu, 27.3c per lb; Zn, 17.5c per lb.

Lead concentrate: 0.471 pct wt; 31,089 g Ag; 167.9 g Au; 45.0 pct Pb; 5.0 pct Cu.

Per Ton Concentrate

	Payments		Deductions			Net	Net Per Ton Ore
Ag	$856.12		Taxes:	$267.97		$588.15	$2.770
Au	179.32		Taxes:	35.68		143.64	0.676
Pb	176.31	= $198.75	Taxes:	51.68	= $62.53	136.22	0.642
Zn	22.44		Treatment:	5.85			
			Freight:	5.00			
			Total net smelter returns, pb conc			$868.01	$4.088

Zinc concentrate (after cyanidation): 0.544 pct wt; 1023.2 g Ag, 5.14 g Au; 50.11 pct Zn.

Per Ton Concentrate

	Payments	Deductions			Net	Net Per Ton Ore
Ag	$22.24	Taxes:	$7.01		$15.23	$0.083
Au	4.21	Taxes:	0.84		3.37	0.018
Pb&Cu		Taxes:	6.00			
Zn	161.33	Taxes:	26.62			
		Treatment:	17.05	= $106.99	54.34	0.296
		Freight:	54.01			
		U. S. Duty	3.31			
		Total net smelter returns, zn conc			$72.94	$0.397

Net bullion returns from cyanidation of zinc conc

	G per Ton Ore Pct Wt x Assay		Cyanide Recovery Pct		Cyanide Recovery G per Ton Ore	
	Ag	Au	Ag	Au	Ag	Au
	55.55	0.2796	90.00	90.00	50.00	0.2516
Gross value per g metal					$ 2.899	$1.1225
Gross value per ton ore					$ 1.452	$0.282
Selling expense (includes taxes)					32.09	21.18 pct
Net value per ton ore					$ 0.986	$0.222
Total net value Ag and Au					$1.208	

Case I

Recoverable from conc by cyanidation

	G per Ton Ore Pct Wt x Assay		Cyanide Extraction Pct		Cyanide Extraction G per Ton Ore	
	Ag	Au	Ag	Au	Ag	Au
Pb conc	183.04	0.9879	89.25	78.80	163.36	0.7785
Zn conc	19.05	0.0820	94.30	96.20	17.96	0.0789
Total extractable by cyanidation					181.32	0.8574
2 pct soluble loss					3.63	0.0171
Net recoverable by cyanidation					177.69	0.8403
Gross value per g metal					$ 2.899	$1.1225
Gross value per ton ore					$ 5.151	$0.943
Selling expenses (includes taxes)					32.09 pct	21.18 pct
Net value per ton ore					$ 3.498	$0.743
Total net value Ag and Au per ton ore					$4.241	

Recap and comparison

	Net Value of Recoveries: Pre-cyanidation Flotation	Net Value of Recoveries: Straight Cyanidation
Ag	$3.728	$3.498
Au	0.896	0.743
Pb and Cu	0.649	—
Zn	0.200	—
Total	$5.473	$4.241

Case II

Recoverable from conc by cyanidation

	G per Ton Ore Pct Wt x Assay		Cyanide Extraction Pct		Cyanide Extraction G per Ton Ore	
	Ag	Au	Ag	Au	Ag	Au
Pb conc	146.43	0.7908	88.50	76.00	129.59	0.6010
Zn conc	55.66	0.2796	92.94	91.70	51.73	0.2564
Total extractable by cyanidation					181.32	0.8574
2 pct soluble loss					3.63	0.0171
Net recoverable by cyanidation					177.69	0.8403
Gross value per g metal					$ 2.899	$1.1225
Gross value per ton ore					$ 5.151	$0.943
Selling expense					32.09	21.18 pct
Net value per ton ore					$ 3.498	$0.743
Total net value Ag and Au per ton ore					$4.241	

Recap and comparison

	Net Value of Recoveries: Pre-cyanidation Flotation	Net Value of Recoveries: Straight Cyanidation
Ag	$3.839	$3.498
Au	0.916	0.743
Pb and Cu	0.642	—
Zn	0.296	—
Total	$5.693	$4.241

Smelter Returns With Pre-Cyanidation Flotation

Evaluation—Flotation vs. Straight Cyanidation

Flotation is followed by cyanidation of the flotation tails, which are first dewatered in three 50-ft diam thickeners. The underflow then joins the underflow of the cyanide plant thickeners and goes to agitation and filtration. The identity of the flotation tails is thus lost, and cyanide plant results on the separate flotation tails are not obtainable.

No entirely satisfactory method has yet been found to compare the net results of the complex flotation with cyanidation. The method used is based on the assumption that the silver and gold extracted by cyanidation of the entire ore would be equal to the sum of the extractions of the flotation products, i.e.,

Increased Net Profit for Pre-Cyanidation Flotation

	Increased Net Returns for Pre-cyanidation Flotation	
	Case I	Case II
From Ag	$0.230	$0.341
Au	0.153	0.173
Pb and Cu	0.649	0.642
Zn	0.200	0.296
Total increased net return	$1.232	$1.452

The increased operating costs are as follows:
For Case I, the operating costs for July 1951 were:

	U. S. Currency, $	
	Per Month	Per Ton Ore
Labor	1,967.66	0.063
Power	1,335.80	0.043
General supplies	1,174.62	0.037
Reagents	2,561.37	0.082
Total	7,039.45	0.225

For Case II, increase the amount for Case I by following estimate of cost of cyanidation of zinc concentrate.
Based on 5.5 ton concentrate from 1000 ton ore per day

	U. S. Currency, $	
	Per Day	Per Ton Ore
Cyanide 20 kg per ton of conc	33.00	0.033
Power	9.00	0.009
Labor	18.00	0.018
Misc.	15.00	0.015
Total	75.00	0.075
Total costs, Case I		0.225
Total costs, Case II		0.300

Net increased profits from pre-cyanidation-flotation are then:

	Case I	Case II
Increased net return	$1.232	$1.452
Increased operating cost	0.225	0.300
Increased net profit	$1.007	$1.152

the lead concentrate, the zinc concentrate and the flotation tailing. This method was accepted after laboratory tests had indicated that the assumption was justified.

In applying the present method of comparison, laboratory cyanide extraction tests are being made on each car lot of lead concentrate and zinc concentrate produced. The samples are treated in strong cyanide solution without further grinding. The same comparison between pre-cyanidation flotation and straight cyanidation was made by comparing the net smelter returns from sale of flotation concentrates with the net bullion value of the silver and gold extracted by the laboratory cyanidation tests. Net increased profits from flotation are then obtained by deducting the operating cost for flotation from the increased net returns as obtained above.

The comparative net returns presented in this paper are based on the method described. In Case I, pre-cyanidation flotation without treatment of the zinc concentrates is compared with straight cyanidation, and in Case II, flotation with zinc concentrate treatment vs. straight cyanidation is shown.

Operating Data

The accompanying flowsheet does not show crushing, weighing, sampling, grinding cyanidation nor tailing disposal which are part of regular cyanide plant operation and equipment.

The grinding is done in two stages; first stage with two No. 86 Marcy ball mills in closed circuit with 8x21-ft Dorr classifiers; the second stage with two 6x10-ft and two 5x10-ft ball mills are used, all in closed circuit with one 16-ft diam bowl classifier.

The bowl classifier overflow (flotation feed) is maintained at a sp gr of 1.20 by a Massco-Adams density controller. This gives 27 pct solids in the pulp and a screen analysis of 8 pct plus 65 mesh and 61 pct −200 mesh.

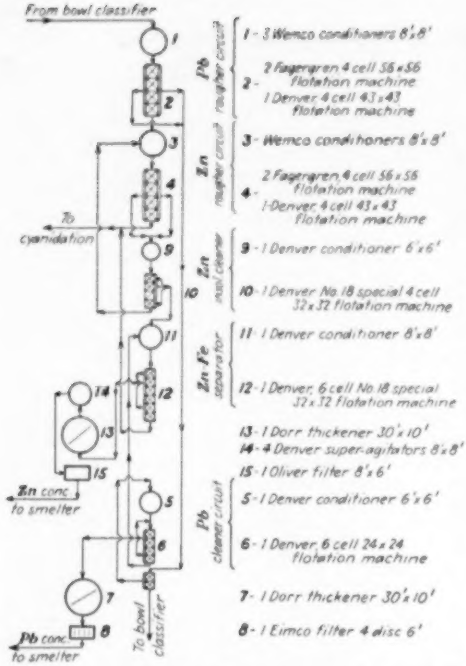
The water used for grinding is composed of about two thirds mine water and one third cyanide regeneration tails solution. This waste solution is used in the flotation plant as a reagent to depress zinc and iron. It is an acid solution, pH 5.4, and carries considerable amounts of sodium chloride, sulphites, zinc salts, and thiocyanate with small amounts of cyanide. Its effectiveness as a reagent is probably due to the zinc salts, sulphites, and cyanide. Before cyanidation of the zinc concentrate is possible, the rougher lead concentrate is cleaned of its insoluable in two Denver Sub-A cells and then sent to thickener and filter. After cleaning insoluble from rougher lead, the concentrate is sent to a 6x6-ft conditioner where lime and cyanide are added and then to the remaining four cells of the 6-cell Denver Sub-A machine.

Returning to description of the primary circuit, from the lead roughers the pulp goes to three 8x8 ft conditioners where copper sulphate, Sodium Aerofloat B and a mixture of 25 pct Aerofloat No. 25 and 75 pct pine oil are added. The pulp then goes to the primary zinc-iron roughers. Reagent No. 301 is added to the third cell of these machines.

The zinc-iron rougher tailings go to cyanidation after being thickened to a sp gr of 1.32 in three 50 ft diam thickeners. The underflow of the thickeners joins the underflow of the cyanide plant thickeners.

The zinc-iron rougher concentrate is cleaned of insoluble by passing through a 6x6 ft conditioner and then a four-cell Denver No. 18 Special. No re-

Smelter Returns Without Pre-Cyanidation Flotation



Addition of the units shown in the flowsheet gave a substantial increase in smelter returns from lead and zinc concentrates. This flowsheet does not show crushing, weighing, sampling, grinding cyanidation nor tailings disposal. These are part of the regular cyanide plant operation.

agents are added and the tails of this machine are returned to the zinc-iron rougher circuit.

After cleaning from insoluble, the zinc-iron concentrate is treated in an 8x8-ft conditioner, adding lime, to give a pH of 11.0, and cyanide in amount of 0.002 kilos per ton of original ore. The concentrates then go to a six-cell Denver No. 18 Special.

The concentrate from these cells is a clean zinc concentrate that goes to a 30-ft thickener. Cyanide and lime are added to the underflow of this thickener and the pulp is placed in four Denver Super-agitators connected in series. The cyanided concentrate is filtered in an Oliver filter and shipped to the smelter. All solution and wash from this filter goes to the cyanide plant pregnant solution.

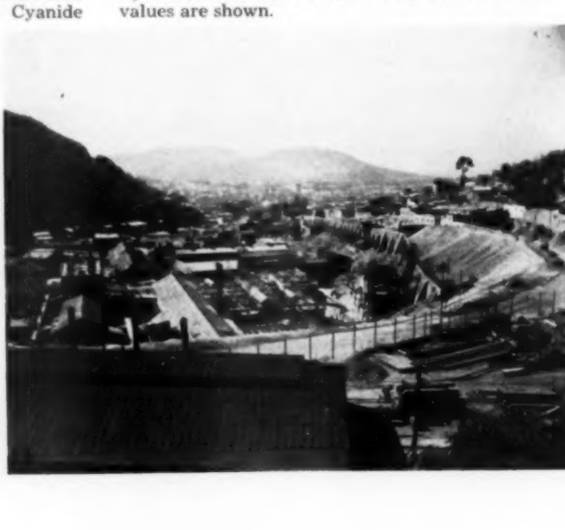
The tails from the zinc-iron separator are an iron concentrate carrying varying amounts of zinc depending upon variations in the ore but generally upon the amount of iron eliminated from the zinc concentrate. The penalty limit for iron in the zinc concentrate is 7 pct and, when allowed to reach this amount, the iron concentrate will assay 35 pct Fe and 10 pct Zn. At present, keeping the iron in the zinc concentrate to less than 4 pct Fe, the iron concentrate assays 25 pct Fe and 20 pct Zn.

The iron concentrate may be disposed of by mixing with the lead concentrate for shipment to smelter or it may be sent to the cyanide plant, depending upon its silver-gold content. At the present time, with assay of 3500 Ag and 15 Au and cyanide extraction of over 85 pct, it is being sent to the cyanide plant.

Eventually it is hoped to use this iron concentrate as a source of sulphur dioxide for the cyanide regeneration plant which now consumes 3 tons of sulphur per day. After roasting this concentrate, the calcine will be mixed with the lead concentrate and shipped to the smelter.

This method of recovering both zinc and iron has, in this case, a number of advantages over the older method that first recovered a relatively iron-free zinc by depressing iron in the primary circuit and then reactivated the iron in the primary to produce the iron concentrate. Both the depression and reactivation in the large-tonnage primary circuits require relatively large amounts of reagents. By using the present method, depression of iron is limited to the small cleaning circuit and no reactivation is required. Another advantage of this method is that cyanide, the most effective iron depressant, is not used in the primary circuit where it might occasion loss of silver by putting it in solution. By limiting the use of cyanide to the small zinc-iron separator circuit where much of the water is recirculated, the chance of silver-gold loss in solution is practically eliminated. Daily assays are made for silver in solution in this circuit and it is very seldom that any values are shown.

View of lower part of Loreto mill showing flotation, agitation, filtration, melting, refining, cyanide regeneration and zinc concentrate cyanidation plants.



Chromium

Ranks Among The Most Strategic of Metals

This is the sixth in the series of articles on strategic minerals: preceding it are "Cobalt" in January 1951, "Sulphur" in May 1951, "Nickel" in August 1951 "Tungsten" in October 1951, and "Lithium" in December 1951.

by Roland D. Parks

THERE are strong indications that chromium steel will one day hold the balance of power among all types of steel. Today, it is going to the forefront in military and civilian use. Stainless steel has received a tremendous impetus because of military demands. Chromium plays a primary, and possibly the most significant roll in the manufacture of stainless and other high temperature alloys. By any standard, chromium ranks among the most strategic of metals. Jet planes would still be on the drawing boards instead of flying in MIG Alley. Many of its more recent applications are the closely guarded secrets of the Pentagon and Government laboratories. Chromium has forged far ahead of the days when it merely decorated an automobile or made the American kitchen sparkle.

The United States consumed one half of all chromium ores produced in the world during the past 50 years and 55 pct of those produced in the past decade. During those same periods, some 85 pct of world production of crude ores came from mines in the eastern hemisphere. Cuba is the only substantial supplier in the western hemisphere and most of the Cuban ore is suitable for use only in refractories.

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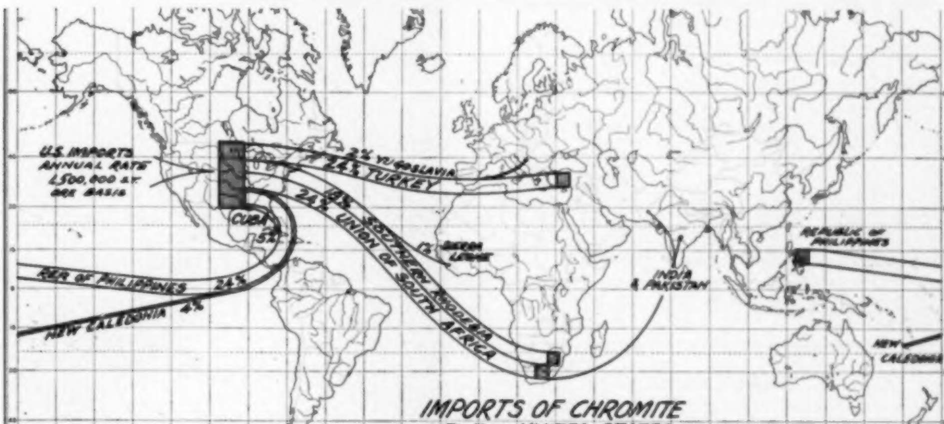
In presenting the strategic aspects of the chromium situation it will be helpful to give certain technical data as background without going beyond the detail needed for the survey.

Use Pattern

Ores containing chromium are consumed by the metallurgical, refractory, and chemical industries. In the United States, postwar consumption by these industries has been in the approximate proportions of 47:37:16, respectively. In two industries, metallurgical and chemical, the ores are first converted to intermediate products.

For metallurgical use, the ores are usually smelted in electric furnaces to ferrochromium, an

The western hemisphere is the major consumer of chromite ores produced in the world, but adds little to this production. Below, world map showing the most important sources of chromite ores.



alloy with iron containing 65 to 72 pct chromium and a maximum of 2 pct silicon. Ferrochromium is added to the bath as required in making alloy steels to specification. Numerous grades of ferrochromium are made by the ferroalloy manufacturers. About two thirds of the production may be classed as low carbon ferrochrome of less than 2 pct carbon content. About one third is high carbon, containing 4 to 7 pct carbon. In recent years chrome ore has come into metallurgical use for production of exothermic chromium alloys, such as Chrom-X, and for making chromium briquets. Briquets contain 2 lb of chromium each and are principally used in ferrous foundries. Roughly 80 to 85 pct of the ore used metallurgically is converted to ferrochromium. Chrome alloy steel in minor amounts is made in the electric furnace from scrap and chromite. Currently, plans are being made to produce electrolytic chromium from ore. Metallurgical uses are given in Table I.

Wider use of chromium is indicated by two recent press announcements; one, the production of chromium carbides of desirable hardness coupled with resistance to oxidation and acid corrosion; and two, more effective control through research of the carbon content of stainless steels promising expansion of high temperature applications.

In the chemical industry sodium bichromate is the primary chemical from which other chromium compounds are made. Principal uses for chrome chemicals, in order of rank, are pigments, tanning, plating, dyes, textiles, and metals. Chromium salts also serve as inhibitors of corrosion when added to aqueous media. The first three account for about 75 pct of the total. Chromium plating, generally considered a metallurgical application, uses the chromium oxide, CrO_3, also called chromic acid, as the source for the metal.

Chromite, the ore of chromium, is a neutral refractory—it is resistant to both acid and basic slags—and because of this rather unique property finds wide application as a furnace lining for areas intermediate between the basic hearth and the acid roof.

Table I. United States Chromium Supply and Consumption[1]
Short tons, all grades

Year	Domestic Production	Imports	Total New Supply	Industry Consumption	Exports
1951	6,897	1,435,069	1,441,966	1,215,386	2,030
1950	404	1,303,713	1,304,117	980,369	2,044
1949	433	1,203,852	1,204,285	672,773	2,382
1948	3,169	1,542,125	1,545,744	875,033	2,894
1947	948	1,106,180	1,107,128	833,357	3,435
1946	4,107	757,391	761,498	734,759	2,158
1945	13,973	914,765	928,738	808,120	12,366
1944	45,629	848,390	894,019	848,449	1,019
1943	160,120	928,576	1,088,696	964,600	26,259
1942	112,876	981,607	1,094,483	891,952	4,743
1941	14,259	1,115,292	1,129,551	800,290	1,906[2]
1940	2,982	736,612	739,594	562,915	n.a.[3]

[1] Data source U. S. Bureau of Mines.
[2] July to December only.
[3] Data not available.

Stainless Steels

Some 30 types of wrought stainless steel are currently being made in the United States. They fall into three groups, according to classification made by Carl A. Zapffe in his book *Stainless Steels:*

1—Hardenable stainless steels; also referred to as straight chromium or martensitic types, where the chromium content of 12 to 17 pct is in the lower stainless range.

2—Nonhardening stainless steels; also referred to as straight chromium or ferritic types, where the chromium content of 18 to 30 pct is in the upper stainless range.

3—Chromium-nickel stainless steels of the 18-8 types; also referred to as austenitic or work-hardening stainless steels. Chromium content for this class ranges from 8 to 30 pct with nickel from 6 to 35 pct.

All stainless steels contain chromium. Nickel is the second major additive in the 18-8 or austenitic types. Carbon, though of low percentage content, is critical in all types.

Class 1 stainless steels, with 12 to 17 pct chromium, resist corrosion at ordinary temperatures. Class 2 types, with 18 to 30 pct chromium, resist oxidation at temperatures to 1000°C in addition to their passivity at ordinary temperature. The nickel content of the Class 3 steels stabilizes the ductile austenitic structure, improves cold working properties, corrosion resistance, and ultimate strength.

In 1948, Class 3 (18-8 types) stainless steels constituted one half of U. S. stainless steel production, Class 2 types, one quarter, and Class 1 types about one eighth.

Chromium Ores

Chromite, or chrome-iron ore, is the only commercial source of chromium. When pure, the mineral chromite contains about 68 pct chromic oxide (Cr_2O_3) and 32 pct ferrous oxide (FeO). It is rarely found in nature in its pure state, since oxides of magnesium and aluminum readily replace those of

Here, molten ferro-alloy is being repoured from ladle into chills. Principal products of Electro Metallurgical's new plant will be alloys of chromium, manganese, and silicon. One product, extra-low-carbon ferrochrome, will be produced here for the first time on a commercial basis. This new ferrochrome product is expected to simplify the production of stainless steel.

iron and chromium isomorphously. Commercial chrome ores seldom contain over 50 pct Cr_2O_3 and some carry up to 20 pct each of MgO and Al_2O_3. Silica may run to 5 pct with lime 2 pct or less.

The mineral chromite is dark colored (brown to black), heavy (sp gr 4.6 for better grades), and relatively soft (Hardness 5.5 on Moh's scale) for a mineral belonging to the spinel group. Its melting point ranges from 1545°C to 1730°C depending on composition. It is almost insoluble in acids.

Commercial Grades of Ore

Chromite does not have a fixed chemical composition. For this reason, certain ores are suitable as sources of chromium for metallurgical and chemical processes while others contain the required combination of elements for refractory use. Commercially, ores are grouped into three classes according to the industries for which their compositions are suitable.

Metallurgical grade ore, for use in the manufacture of ferrochrome, should contain a minimum of 48 pct Cr_2O_3 with a chromium-to-iron ratio of not less than 3 to 1. In addition, a hard, lumpy structure is usually called for with pieces ranging from 6 to ½ in. in size. Silica is undesirable and combined magnesia and alumina of over 25 pct may be objectionable. In practice, ores somewhat below these standards either are used directly or blended with higher quality material.

Refractory grade chromite usually contains about 63 pct combined Cr_2O_3 and Al_2O_3 with 57 pct a common minimum. Typical ores contain about 34 pct Cr_2O_3. Iron and silica should be low, usually around 10 and 5 pct, respectively. Hard lump ore is desirable for making bricks, and ground material is suitable for cement. Magnesia content ranges around 15 pct.

Chemical grade chromite should contain a minimum of 45 pct Cr_2O_3. High iron content is not harmful within reasonable limits. A common chrome-to-iron ratio is 1:6:1. Silica must be less than 8 pct and sulphur low. Fines and concentrates are preferred because they disintegrate readily in processing.

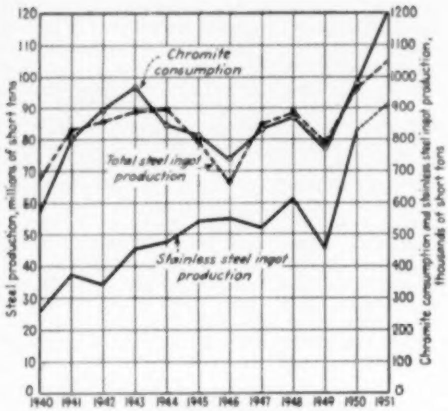
Ore Deposits

Chromite deposits, usually resulting from magmatic segregations, are found in ultrabasic igneous rocks, such as peridotites and their alteration products, commonly serpentines. Deposits are most fre-

World Production of Chromite, Principal Countries Only, 1941-50, in Metric Tons

Country	1941	1942	1943	1944	1945	1946	1947	1948	1949	1950
NORTH AMERICA										
Canada	2,152	10,393	26,848	24,543	5,221	2,821	1,961	1,536	327	—
Cuba	163,175	286,470	354,152	192,131	172,626	174,350	159,209	118,624	97,368	[5]65,817
Guatemala	697	529	374	97	443	610	625	474	300	300
United States	12,935	102,400	145,259	41,394	12,676	3,726	860	3,283	393	367
SOUTH AMERICA										
Brazil (exports)	5,944	5,776	7,813	4,721	1,490	174	—	1,626	3	—
EUROPE[6]										
Albania	20,000	37,797	31,091	—	—	—	—	16,500	[1]	[1]
Bulgaria	[1]	5,000	5,000	5,000			[1]	[1]	[1]	[1]
Greece	16,240	24,300	15,500	18,295	2,413	9,062	2,640	1,500	3,381	12,631
Portugal	—	1,275	1,267	1,111	1,669	1,530	533	440	[2]500	[2]500
Yugoslavia[2]	[1]	100,000	65,000	10,000	6,000	68,000	55,000	65,000	83,000	100,000
ASIA										
Cyprus (exports)	4,816	2,936	7,986	469	1,070	1,158	5,283	6,899	14,875	18,440
French Indo-China	—	3,570	6,510	2,300	—	—	—	—	—	
India	50,940	50,380	33,789	40,190	31,642	45,511	35,274	22,917	19,728	[2]20,000
Japan	54,519	67,540	58,520	71,135	28,539	7,079	2,407	9,340	27,003	31,953
Pakistan	[4]	[4]	[4]	[4]	[4]	[4]	22,040	18,160	15,925	[2]18,000
Philippines	[8]125,343	[3]50,000	[3]60,000	[3]70,000	[3]50,000	58,930	196,185	250,854	246,744	250,511
Turkey	135,714	116,342	154,512	182,108	146,718	103,167	102,875	285,353	434,117	350,000
U.S.S.R.[2,6]	[1]	400,000	325,000	300,000	300,000	300,000	300,000	600,000	350,000	500,000
AFRICA										
Sierra Leone	13,907	16,726	16,306	9,851	578	10,301	16,769	7,886	22,101	7,517
Southern Rhodesia	322,123	348,314	287,453	277,051	186,318	151,433	155,149	230,703	243,506	291,525
Union of South Africa	141,884	337,620	163,232	88,909	99,080	212,253	373,094	412,783	404,351	496,324
OCEANIA										
New Caledonia	64,309	67,610	46,952	55,229	50,838	24,946	50,530	75,021	88,792	[2]90,000
WORLD TOTAL (ESTIMATE)[7]	1,752,000	2,031,000	1,825,000	1,400,000	1,100,000	1,200,000	1,700,000	2,150,000	2,100,000	2,300,000

[1] Data not available; estimate included in total.
[2] Estimate.
[3] Planned production, as reported.
[4] Included with India.
[5] January to October, inclusive.
[6] Output of U.S.S.R. in Europe included with U.S.S.R. in Asia.
[7] In addition to the countries listed, totals include production when available, and estimates for Afghanistan, Argentina, Australia, Egypt, Iran, Mexico, Sweden, United Kingdom.
[8] Exports reported as 117,358.



Comparison of chromite consumption, total steel production, and stainless steel production in the United States for the period, 1940-1951 inclusive.

quently pouch-shaped lenses or tabular masses but may also occur as regular parallel (stratiform) seams, as residual surface concentrations from chromite disseminated throughout the country rock, and often as alluvial sands.

Although individual chromite desposits rarely contain more than 50,000 tons, a few deposits are much larger and constitute in themselves the bulk of the known world reserves. Most extensive deposits known, though not everywhere of the highest grade, are those in Southern Rhodesia and in the Transvaal, Union of South Africa.

Southern Rhodesia

The Great Dyke series of ultrabasic rocks cut northeasterly across Southern Rhodesia in nearly a straight line for more than 300 miles. The ultrabasic formations are from 3 to 6 miles wide and the chrome seams dip flatly inward from both sides of a syncline. Two to seven parallel chromite bearing seams, from 1 to 10 in. in thickness are found at vertical intervals of 30 to 200 ft in serpentine members near the base of the series. These seams can be traced on the surface for many miles and in several areas have been mined completely through from one side of the Dyke to the other. It has been estimated that the seams could contain mineable reserves of 125,000 to 200,000 tons per mile to a depth of 100 ft down the dip, or an estimated 42 to 66 million tons.

At Selukwe, in the center of the country, massive deposits of chromite are found in talc schist formations a few miles off the course of the Great Dyke. These deposits of high grade, lumpy ore, generally of metallurgical grade, though not as extensive as the Dyke formations, are more readily mineable and have constituted the mainstay of Rhodesian production for many years. In recent times, the Dyke deposits, principally in the Umvukwe district, have increased in importance.

Table II. Metallurgical Uses of Chromium

Major	Approx Pct Industry Use
Stainless Steels:	60
Austenitic type—18 Cr, 8 Ni	
Martensitic and ferritic types—12 to 27 pct Cr	
Low Alloy Steels:	15 to 20
SAE 4140 type—½ to 1 pct Cr	
High Temperature Alloys:	10
15 to 25 pct Cr combined with similar amounts of cobalt, nickel, etc., and iron	
High Speed Steels:	5 to 10
1 to 4 pct Cr—general type: 18 W, 4 Cr, 1 V	
Minor	
Dental and Surgical Alloys:	1
20 to 25 pct Cr—Alloys such as Vitallium containing approx 65 Co, 26 Cr, 6 Mo, 2 Ni	
Chromizing	
Other Alloys	
Allied	
Chromium Plating:	2 to 3
Considered a metallurgical use but derives its Cr from chromic oxide, CrO_3, a chemical industry product	

Union of South Africa

In the Transvaal, chromite occurs in two extensive areas in the gabbro zone of the Bushveld complex. The ore, granular chromite in silicate gangue, is found in closely spaced parallel seams conformable with the surrounding basic rocks and dipping flatly toward the center of the basin. Overall structure and occurrence are similar to the Great Dyke deposits in Southern Rhodesia.

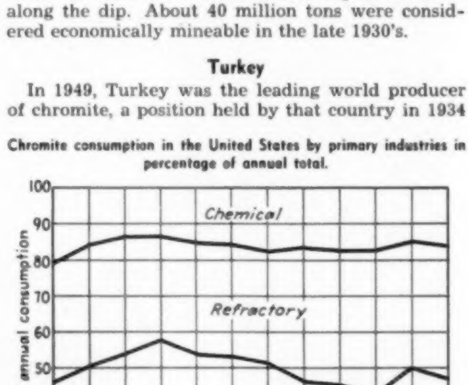
Seams vary in thickness from 1 in. or less to more than 6 ft. Two groups of seams are known in the Lydenburg or eastern belt and three groups in the Rustenburg or western belt. The crescent shaped chrome-bearing formations extend 70 miles in the eastern belt and 100 miles in the western belt. The eastern belt has been more productive. Since the chrome-iron ratio is generally about 2 to 1 for these ores, the friable grades containing 47 to 49 pct chromic oxide are sold for chemical rather than metallurgical use. The hard, lumpy ores, of about 46 pct chromic oxide content, are used mainly for refractories because of desirable compositions of alumina and magnesia.

Total reserves of the Transvaal have been estimated at some 200 million tons to depths of 500 ft along the dip. About 40 million tons were considered economically mineable in the late 1930's.

Turkey

In 1949, Turkey was the leading world producer of chromite, a position held by that country in 1934

Chromite consumption in the United States by primary industries in percentage of annual total.



Heavy user of chromium alloys, is this B-47 Stratojet built by the Boeing Airplane Co. for the Air Forces. The military importance of air power has put chromium high on the list of critical materials.

and for many years prior to 1900. This revival of Turkey's most important mining industry is due in no small part to recent changes in the mining laws of the country making chromite mining attractive to both domestic and foreign capital. Turkish chromite is of metallurgical grade. Much of it is high grade lump ore of the most desirable quality.

Chromite deposits are widespread throughout the country, some 120 known prior to World War II, but in many cases remote from transportation. Some of the deposits are veins. Others are typical tabular masses. The largest deposits are in the Guleman district near Ergani in eastern Turkey, where reserves of high grade ore were estimated some years ago at over 1 million tons for the Guleman mines. Other sizable deposits are nearby. The Kutayha region west of Ankara in which the Dagardi mine is located, and Fethiye mines on the southwest coast, have long been prominent. The Eskisehir district more recently became important due to the Karvek mine, now second largest producer in Turkey. Important new deposits have been reported in the Taurus Mountains in south central Turkey.

Past estimates of reserves for Turkey, ranging from 3 to 15 million tons, appear now to have been low. Reserve estimates will probably be increased through exploration and development.

Republic of the Philippines

Two chromite deposit belts are known in the Philippines. They are the western belt extending down the western sides of Luzon, Mindoro, and Panay Islands to north central Mindanao, and the eastern belt from southeastern Luzon through Samar and Dinogat Islands to northeastern Mindanao.

Five types of deposits have been recognized, ranging from large solid masses, through lenses of varying sizes to boulders, disseminated and placer deposits. Most prominent is the massive deposit at Masinloc in Zambales province, at the northern end of the western belt in Luzon. This unique deposit of refractory grade ore assaying about 34 pct chromic oxide, is estimated to contain proved reserves of 10 million metric tons, or about 90 pct of the total measured reserve of the Islands. Several of the smaller deposits contain chromite of chemical and metallurgical grade.

New Caledonia

At Mount Tiebaghi, near the northern end of New Caledonia, chromite is found in veins and funnel shaped columns extending deep into the peridotites and serpentines of the mountain mass. The Tiebaghi mine, largest producer in New Caledonia, is famous for its high grade metallurgical ore. Chrome-bearing sand deposits are found in the west coast and

Table III. Total Production of Chromite by Ranking Countries, From 1906 Through 1950[1]

Country	Total Production Metric Tons.	World Total, Pct	Ore Produced, Principal Classes[2]
Southern Rhodesia	5,949,000	18.6	M R
USSR	4,900,000	15.3	M C
Union of South Africa	3,960,577	12.4	M C
Turkey	3,658,000	11.4	M
Cuba	2,507,051	7.8	R
New Caledonia	2,303,000	7.2	M
Philippines	2,044,000	6.4	R
India	1,465,000	4.6	M
Yugoslavia	1,223,000	3.8	M
Japan	714,456	2.2	
Greece	690,057	2.2	
United States	527,750	1.6	
World Total	32,000,000±	93.5	

[1] Based on Dept. of Commerce data supplemented by estimates where firm data not available.
[2] M, metallurgical; R, refractory; C, chemical.

other ores requiring concentration are worked near Noumea in the southwest part of the island.

There is little information available on reserves at Mount Tiebaghi. They were estimated in 1931 at 1 to 1½ million tons of ore containing over 50 pct chromic oxide. Since then over a million tons have been produced and output is now at a peak rate. The Fantoche mine in the Tiebaghi area was reported some years ago to be the deepest underground chromite mine in the world.

Cuba

As a western hemisphere producer, mainly of refractory ore, Cuba was called upon so heavily during the last war that exploration is needed to restore capacity. Although favorable zones have been noted along much of the north coast, production is limited almost entirely to Camaguey and Oriente Provinces at the east end of the island. The irregular sackform deposits range in size from a few tons to over 100,000 tons and appear to be distributed at random in the chromite-bearing zones. Geologic structures in the Moa district in Eastern Oriente Province have recently been the subject of detailed study.

United States

The largest known chromite deposits in the United States are in the Stillwater complex in south central Montana. This area of ultra basic rocks, varying from 1 to 5 miles wide, extends along the north flank of the Beartooth mountain range for about 30 miles and contains a number of chromite deposits. Benbow and Mouat-Sampson at the eastearn end are considered the most important.

During World War II, both Benbow and Mouat were brought into production after construction of mining and milling plants, townsites, and aerial tramways. In all, some 400,000 tons of ore were mined, of which about 250,000 tons were milled to produce concentrates. At Benbow, the average Cr_2O_3 content of the ore, 18.4 pct, was improved to about 41.5 pct in the mill concentrates. At Mouat, an average head of 19.3 pct Cr_2O_3 was raised to 38.8 pct by milling. An analysis of the ore at Benbow indicates that it contains only about two thirds as much chromium as pure chromite and for this reason the Cr_2O_3 content cannot be beneficiated above 46 pct. The average chrome-iron ratio varies from about 1.5:1 for the better ores to about 1.7:1 for the concentrates. Though not up to standard specifications for metallurgical grade ores, the Stillwater products are considered usable. It is understood that arrangements are being made by American Chrome Co. to reopen the Mouat property.

Reserves in the Stillwater area, as reported by the U. S. Geological Survey and U. S. Bureau of Mines in a recent publication on the mineral resources of the United States, are about 1 million long tons of contained Cr_2O_3 in measured and indicated ore, plus an additional ½ million long tons of oxide in inferred ores. This total is thought to represent about 75 pct of known domestic resources.

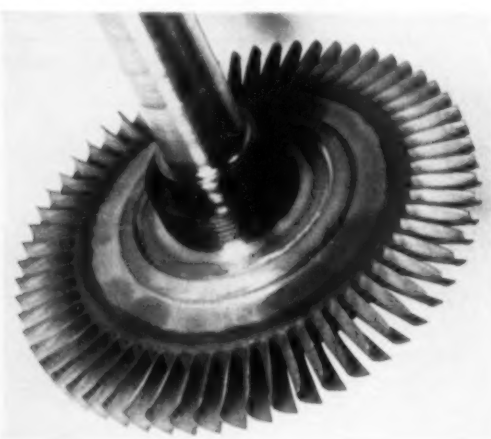
Beach sands in southwestern Oregon and numerous lens deposits in the mountain ranges of California account for most of the balance of domestic reserves. Many of these deposits have a higher Cr_2O_3 content and better chrome-iron ratio than the larger deposits at Stillwater and much of the current mining activity is centered in this area. Other deposits of some prominence are located in Wyoming and on the Kenai peninsula, Alaska.

The estimated total domestic reserve of some 2 million tons of contained chromic oxide, while perhaps on the conservative side, points up clearly the need for maintaining reliable imports from the major foreign sources.

Government Buying Program

To stimulate domestic production, the United States Government has recently reopened the ore buying station at Grants Pass, Ore. The program calls for purchase for stockpile of up to 200,000 long tons of specification chrome ores or concentrates from domestic miners in lots of 5 tons or more. Not over 2000 tons per year may be accepted from a single producer without prior arrangement.

Specifications permit purchase of lump ores, fines, or concentrates carrying 42 pct or more of chromic oxide, 10 pct or less of silica, and having a minimum chromium-to-iron ratio of 2 to 1. Premiums will be paid according to the published schedule for chromic oxide content above 48 pct and for chrome-iron ratios above 3 to 1 but not exceeding 3.5:1. Correspondingly, penalties are scheduled for chromic oxide contents from 48 down to 42 pct and for chrome-iron ratios in the range from 3 to 1 down to 2 to 1.



Compressor rotor disc from a Wright Turbo Jet engine is made up of alloys with chromium content ranging from less than one to more than 12 pct. Alloys with suitable creep and fatigue characteristics at elevated temperatures are necessary because of blade vibration, bending, and centrifugal loads on the rotor blades. Conventional reciprocating engines are consumers of chromium alloys.

The growth of the defenses of the Western nations is a heavy drain on the chromium supplies. A new tank, mounting a 75 mm gun, has innumerable uses for the stainless alloys, of which chromium is a vital component.

Base prices for purchases are $110 and $115 per long dry ton delivered at the station, or more than double current quoted prices for similar ores from foreign sources at Atlantic seaboard.

Although the program has been in effect only a few months, there is indication that domestic output is already being expanded. Production for the last quarter of 1951 was 5600 tons as compared with 1300 tons for the first three quarters of the year.

World Reserves

World reserves of commercial chromite, though not measured accurately, appear at this time to approximate 300 million tons of ore of all classes, metallurgical, refractory, and chemical.

No attempt has been made in preparing this summary to learn the reserve data of countries within the Russian sphere because there is a general lack of published information for iron curtain nations. It is assumed that field investigation has increased their resources in recent years, similar to the trend in the free world countries. However, the general magnitude of world reserves would not appear to be changed by this lack of data.

Presently known reserves are found largely in five countries, of which two are outstanding. The approximate size of reserves in these and other prominent producing countries, based on published and reported information, are shown in Table IV.

Table IV. World Reserves of Chromium

Major Reserves	Magnitude, Millions of Tons	
Union of South Africa	150±	
Southern Rhodesia	50±	
Turkey	20±	
Republic of Philippines*	20±	
USSR	15	(1937)
Secondary Reserves		
New Caledonia*	5±	
United States	5±	
Cuba*	5±	
Yugoslavia, India, Pakistan	5±	

* Not including extensive deposits of lateritic iron ores containing about 1.5 pct chromium.

In the preparation of the preceding table, the usual standards of ore estimation involving recoveries, grades, and cost-price relationships, have obviously been relaxed in the interest of arriving at a comparative scale of long range availability of chromite from sources throughout the world.

Future Demand

It is interesting to speculate on the future of the general raw material situation and for a metal like chromium specifically, when its properties are so essential to the making of new alloys. As stated, the end uses of chromium are largely associated with steel. And steel in turn, being a cornerstone of the economy, is associated rather directly with the needs of all people. To crystal gaze for the future of chromium look to the past as a guide to the future.

Census data show the population of the United States doubled between 1900 and 1950, an increase of 1⅜ pct per year. During the same period, steel production quadrupled, increasing about 1½ pct per year, considered on a per capita basis. If both rates continue until 1960 the indicated steel production would then be one third more than the 1950 output. Though oversimplified, the hypothesis illustrates the severity of future mineral requirements.

A close relationship has existed in the United States in recent years between chromite consumption and steel production, together with the advancing output of stainless steels. As long as these conditions hold and as chromium becomes more important in the manufacture of high temperature alloys the demand for chromium steel may be expected to exceed the general trend for all types of steel. On the basis of these trends, annual consumption of chromite should reach 1 million short tons by 1960.

Since the pattern of the strategic nature of chromium seems well established for today and tomorrow, it would be well to be constantly mindful of our dependency on foreign sources for the raw material of this metal. To this end there is the insurance of a stockpiling program carried on with vigor and foresight. More important is the need for constancy of trade relations with suppliers, wise investment policies, and political stability.

Acknowledgment

In preparing this brief survey of the chromium situation the author acknowledges gratefully the many observations on current activities that have been given to him personally and the published sources from which he has drawn freely, in particular, the reports of the U. S. Depts. of Interior and Commerce.

Books for Engineers

Wire Ropes in Mines. Proceedings of a conference held at Ashorne Hill, Leamington Spa, Warwickshire, September, 1950. Published by the Institution of Mining and Metallurgy, Salisbury House, London, 1951. 828 P. $7.00.—The volume contains 18 pages on the manufacture and properties of rope wire and wire ropes. In addition, wire rope practice in Britain, Ontario, the Witwatersrand, The United States, South Africa, and other countries are discussed. Testing, government regulations, and inspection are also treated. The book contains a three day discussion on these subjects complete with a summary of conclusions and recommendations.

Heating Ventilating Air Conditioning Guide 1952. Published by the American Society of Heating and Ventilating Engineers, New York, 1952. 1496 P. $7.50.—Contains a technical data section of reference material on the design and specification of heating, ventilating and air conditioning systems. Also has a section containing information on equipment and manufacturers. An index to technical and equipment sections is included.

Principles of Geochemistry. By Brian Mason. Published by John Wiley & Sons, Inc., New York and Chapman & Hall Ltd., London. 276 P. $5.00.—The book attempts to state the facts of geochemical experimentation in principles easily mastered by the student. The mass of facts is synthesized into a coherent description of the physical and chemical evolution of the earth. Questions such as the geochemistry of igneous, sedimentary and metamorphic rocks, the origin and evolution of the ocean, the role or organisms in the concentration and deposition of individual elements and the nature of the primeval atmosphere are discussed in detail.

Theoretical Petrology. By Tom F. W. Barth. Published by John Wiley & Sons. 387, $6.50.—In this valuable book Dr. Barth gives the present status of our knowledge in all fields of petrology covering igneous, sedimentary, and metamorphic rocks. Theoretical Petrology is an exact and comprehensive scientific study of the rock making processes, and an interpretation and synthesis of the existing quantitative data—the field data from all over the world, the experimental data from geophysical and geochemical laboratories, and theoretical data of the research workers in geology, chemistry and physics. From this data—collected, organized and interpreted, Dr. Barth sketches the cyclic history of the rock masses of the earth's crust: their origin, evolution, transfiguration, destruction and redevelopment.

Introduction To Geophysical Prospecting. By Milton B. Dobrin. Published by McGraw-Hill Book Co. 435, $7.—Elementary in approach and requiring no mathematics beyond trigonometry, this text presents the fundamental principles and methods of geophysical prospecting. All standard methods are treated, and for each method the following aspects are covered: fundamental physical principles, instrumentation, field techniques, and reduction of field data and interpretation of the data. Emphasis is placed on the information each method yields and its most likely application in exploration. Gravitational, magnetic, seismic, electrical, and radioactivity prospecting are treated, as well as logging. Three chapters are devoted to such topics from pure geophysics relating to prospecting as earthquake seismology, terrestrial magnetism, and the gravitational field of the earth.

Sources of Iron Ore in Asia. By Joseph F. Harrington and Benjamin M. Page. Published by General Headquarters, Supreme Commander for the Allied Powers, 1952. 176 P.—The resources of each Asian country, its present state of development and what can be expected in the future, are covered in the book. The book highlights the importance of the Japanese iron and steel industry, but points out that in the past most iron mines in the neighboring countries were primarily developed to feed the conquerors. The book is largely nontechnical, and is intended for the average reader, although the engineer will gain a widened horizon of information on the world metal situation from it.

Injury Experience in Coal Mining, 1948. By Forrest T. Moyer, G. D. Jones, and V. E. Wrenn. Published by the U. S. Bureau of Mines, 1952. 109 P.—The book contains a detailed analysis of factors influencing mine safety and related employment, production, and productivity data. The book is presented in three sections, complete with tables constituting a subsection. The tables contain statistics on number of injuries, frequency and severity, detailed and major cause of injuries, records of coal-producing states, and other data. A great amount of space is given over to discussion of coal mine injuries in Pennsylvania. The third section contains a list of major coal mine disasters.

The Welding of Non-Ferrous Metals. By E. G. West. Published by John Wiley & Sons, Inc., New York. 553 P. $8.50.—The book was written with the intention of filling the need for information which until now has been difficult to obtain, as little was done before 1930 in the welding of non-ferrous metals. The book has two main classes of reader in mind. One is the welding engineer, welding operator, welding instructor and the trainee, the other is the designer, works engineer and metallurgist. No attempt is made to substitute for actual experience. A description of the process and equipment of welding is included for the benefit of the reader who is new to welding. References to original papers have been included for those who wish to go beyond the scope of the work.

Cottrell—Samaritan of Science. By Frank Cameron. Published by Doubleday & Co., $4.50.—Frederick Gardner Cottrell was one of the leading scientific figures of the present century. Yet he has remained almost unknown to the layman, despite an enormously productive life, because of his deliberate policy of self-effacement. His many patents—and he could have cashed in on a thousand ideas which he gave away—were turned over to a nonprofit foundation for the advancement of science. From early boyhood Cottrell emulated those whom he called "men of intensity"—men whose furious pace of work enabled them not only to keep up with the tremendous surge of technological progress, but also to make a positive contribution to it, and to human welfare. His own work was mainly concerned with electrical precipitation of smoke, dust, and fume and the recovery of valuable materials from them.

Geologic Guidebook of the San Francisco Bay Counties, Bulletin 154 prepared under the direction of Olaf P. Jenkins, Chief of the Division of Mines, and issued by Division of Mines, Ferry Bldg., San Francisco, Calif. $2.50—This volume contains a sectional geologic map covering twelve counties, a series of geologic travel logs, and a large number of separate articles which cover a broad scope of subjects—history, science and natural resources, all written by authorities in their subjects. The book is profusely illustrated with photographs, tables, charts, and maps and is a storehouse for information concerning earth sciences. Part I covers Opening of the Golden Gate and the historical background of this area. Part II covers history of the landscape. Part V covers utilization of minerals in industries of the San Francisco Bay Counties and magnesite mineralization in the Red Mountain District.

Geology of The Hayden Creek Lead Mine, Southeast Missouri

by Ernest L. Ohle

Fig. 1—Index map of Missouri showing the location of the two producing areas in the Southeast Missouri Lead Belt and the position of the Hayden Creek Mine (asterisk).

The newly opened Hayden Creek lead mine represents a variation from the usual Southeast Missouri type. Galena mineralization occurs in sandy dolomite cementing a conglomerate of granite boulders and in cracks in the boulders. The conglomerate occurs at the base of the Bonneterre formation on the slopes of two buried pre-Cambrian knobs.

IN 1943 diamond drilling from the surface in an area 2 miles southwest of Leadwood, Mo., discovered a lead deposit of a markedly different character from the usual southeast Missouri type. Subsequent drilling of about 500 holes indicated several million tons of ore, and in 1950 a 701-ft vertical shaft was sunk into the orebody. This paper describes the geology of this interesting occurrence insofar as it is known from the drilling and the small underground exposure at the bottom of the shaft.

Geology

The location of the Southeast Missouri Lead Belt and of the Hayden Creek property are shown in Fig. 1. As shown here, there are two separate producing areas, one from which the larger tonnage has come, located near Bonne Terre, Flat River, Desloge, and Leadwood, and the other to the south around Fredericktown and including famous old Mine LaMotte.

The best published description of Lead Belt geology is by Tarr.* In pre-Cambrian time in this area, an assemblage of granites and rhyolitic flow rocks was eroded to a mature topography. Sedimentation, beginning in the Upper Cambrian, filled in the valleys and gradually covered even the highest hills. The first formation deposited was the Lamotte sandstone; it is successively overlain by the Bonneterre dolomite, the principal lead-producing formation, then the Davis shale, the Derby-Doerun limestone, and a succession of Cambro-Ordovician cherty dolomites. Several of these formations may be found in direct contact with the pre-Cambrian as their deposition progressively overlapped the older rocks on the flanks of the igneous knobs and ridges.

Immediately overlying the pre-Cambrian there is, not uncommonly, a thin basal conglomerate of granite or porphyry boulders, which is incorporated into the overlying sediments. Deposition of the lower part of the rock section was fairly continuous, and the Lamotte sometimes grades almost imperceptibly into the lower sandy Bonneterre. Since the sediments were deposited on an uneven floor, they tend to reflect the irregularities in that floor. Most of the steep dips found in the district, that is, dips in excess of 10°, are original dips and not caused by deformation. Such domelike structures in the sedimentary rocks around the igneous knobs have localized a considerable number of the orebodies in the district and particularly those in the Fredericktown area. Many of the larger orebodies, however, have no close relation to the pre-Cambrian topography. Most of the lead production has come from the lower 150 ft of the Bonneterre formation, where disseminated ore spreads laterally parallel to the bedding for great distances. Average stoping height is about 20 ft, but in a few places there are stopes over 175 ft high.

* W. A. Tarr: Origin of the Southeastern Missouri Lead Deposits. Economic Geology (1936) 31, Nos. 7 and 8, pp. 712-734, 832-866.

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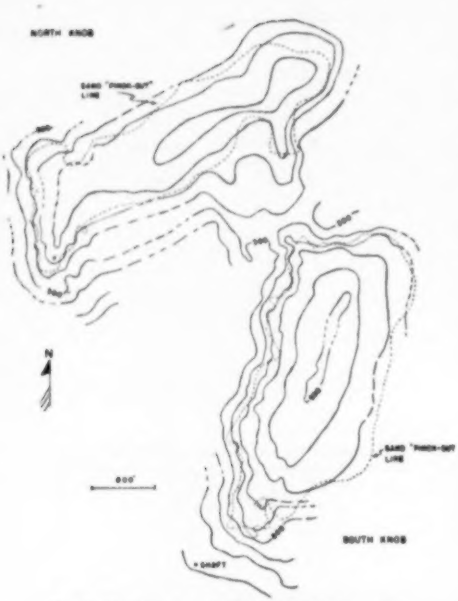

Fig. 2—Configuration of the pre-Cambrian granite surface in the Hayden Creek area. The dotted line indicates the position of the sand pinch-out.

Fairly strong faulting has occurred in southeast Missouri and the main Lead Belt area is surrounded by three faults with displacements ranging from 100 to 600 ft. The 600-ft down drop of the north side of the Simms Mountain fault has resulted in a reversal of the regional dip and the rocks in the Flat River area now dip 1° to 2° southwest.

Hayden Creek Mine

The orebodies of the Hayden Creek area are related to highs in the pre-Cambrian surface. Fig. 2 shows the configuration of the top of the granite as known from drillholes. As shown here, there are two elongated pre-Cambrian hills in the area that gave rise to domal structures in the overlying sediments. These may be designated as the South Knob and the North Knob.

These two prominences are separated by a deep saddle that joins the north end of the South Knob and a point midway along the south side of the North Knob. Lamotte sand is present in this saddle. The line along which the sandstone feathers out up the granite slopes, locally called the "sand pinch-out line," is indicated in Fig. 2. It will be noted that the sandstone rises 50 to 100 ft higher on the northeast slopes than on the southwest slopes of both knobs. This is believed to indicate a prevailing direction of current movement from southwest to northeast during Lamotte time. Other knobs in the Lead Belt show a similar relationship.

Granite is not exposed here at the surface, but the dome structures are reflected in the Davis and Derby-Doerun formations outcropping in the area. The knobs extend up into the lower Bonneterre, but upper Bonneterre beds are present over the highest points. Drillholes show that the area had over 400 ft of relief in pre-Cambrian time.

The South Knob is roughly 4000 ft long by 2000 ft wide, with the long axis trending slightly east of north. The ridge line is sharply defined and the sides have a fairly constant angle of slope from the crest line to the lowest known points where the granite is covered by Lamotte sandstone. The west flank is slightly steeper than the east flank and has an average slope of 20 pct. The largest known body of ore lies on the southwest end of this knob.

The North Knob trends about N 60° E and has approximately the same lateral dimensions as the South Knob. However, it is not as high. Consequently the slopes are more gentle and the crest is not as sharp. Here again, the western slopes are the steepest. The known orebodies associated with this knob are along this western slope.

The interesting thing about the Hayden Creek orebodies is that the ore, instead of being in dolomite, is in a conglomerate of granite pebbles and boulders. Ore in boulders has been mined or is known to exist at several other places in the district, both in the main Lead Belt and in the Fredericktown area. In these bodies, however, the tonnages have been small, ranging from a few hundred to a few tens of thousands of tons and, further, their geological history has been relatively simple. The unusual character of the host rock and a unique stratigraphic setting make the Hayden Creek orebodies of more than passing interest.

Hayden Creek Conglomerate

Part of the conglomerate at Hayden Creek is the common 2 to 20 ft thick basal boulder layer that directly overlies the solid granite hill and was covered by deposition of the Lamotte and Bonneterre formations, a relationship that can be found in many places in the central Ozark region. However, the largest bodies of conglomerate and the ones containing most of the ore had a more complex history. These bodies lie not beneath the Lamotte in the

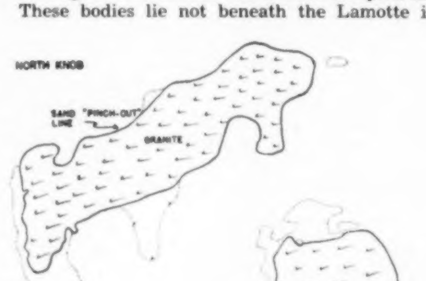

Fig. 3—Location of the bodies of upper conglomerate around the North and South knobs.

normal basal conglomerate position but on top of it and separating it from the overlying carbonate rocks of the Bonneterre. Thus, in selected places, drillholes penetrate two boulder zones, one on top of the Lamotte and the other beneath it. This situation has not been recognized outside of the Hayden Creek area.

Fig. 3 shows the location of the bodies of upper conglomerate around the North and South Knobs, the outline of the knobs being indicated by the sand "pinch-out" lines. It is at once apparent that the largest concentrations of boulders are on the west sides of the two knobs and that both knobs have elongated tongues or tails of boulders that extend southward out over the top of the Lamotte for hundreds of feet. In the case of the South Knob, the boulder tongue is 1200 ft long. It is this boulder mass which contains the bulk of the ore in the Hayden Creek area and in which No. 22 mine is located.

Fig. 4 shows the conglomerate associated with the South Knob in greater detail and also indicates the position of the two cross-sections reproduced in Figs. 5 and 6. These two cross-sections indicate the interrelations of the boulder beds, the Lamotte sand, and the underlying granite hill. Where the sand wedges out up the slope, the upper and lower conglomerate beds cannot be distinguished, and there is a continuous boulder layer from the base of the sand-free Bonneterre dolomite to the top of the granite, except for thin beds or pockets of sandy dolomite that are found locally in the conglomerate. In many places there is no sandy dolomite in the lower Bonneterre, other than as cement between boulders, and the conglomerate occupies all of the position where the sandy transition beds normally would be found.

It will be noted in Figs. 5 and 6 that the angle of slope of the Lamotte surface flattens to the westward where the Lamotte has filled in the pre-Cambrian valley. Section BB′ particularly shows that near the toe of the conglomerate the slope actually reverses itself. This reversal must have opposed the tendency of the main boulder mass to migrate further westward. The front of the conglomerate is quite steep and in one place the thickness decreases from 110 ft to zero between two holes 100 ft apart.

Fig. 7 shows the main South Knob boulder pile in greater detail. By contours on the top of the Lamotte, that is, the base of the upper boulder bed where it is present, it shows the westward and southwestward slope under most of the conglomerate. The reversal in slope indicated before in section BB′ is shown to be an embayment in the contours so that the boulders seem to have been trapped in a small pocket on the sand floor.

Fig. 8 shows the third dimension of this boulder body by thickness contours. Several drillholes cut over 100 ft of boulders. The contours indicate a slightly asymmetric distribution of the boulders with the greater concentration, and hence the steeper marginal slope, occurring on the southeast side of the mass. The edge shows a sharp drop-off everywhere; only two holes were so located near the edge that they cut less than 10 ft of boulders and only five holes cut less than 20 ft. The thickness of the average intersection was 60 ft. In general the axis of this boulder body is parallel to the axis of the South Knob.

The main boulder mass associated with the North Knob is a tongue extending out over the Lamotte for 900 ft in a S 45° E direction. Thus it is elongated

Fig. 4—Plan of the upper conglomerate associated with the South knob.

nearly at right angles to the long axis of the knob. The average thickness of boulders in this body as shown by the drillholes is only 20 ft and the total volume of boulders is less than one eighth of the volume of the big mass near No. 22 shaft. The maximum thickness cut was 40 ft. From observation of the drill cores, it is believed that the average size of the boulders here is smaller than in the main South Knob orebody.

The granite boulders and fragments of the Hayden Creek area range in size from about a millimeter to several feet. The larger ones, 6 in. and over, are almost all well rounded; the smaller fragments are markedly angular and apparently are the result of sharp impacts of the big boulders against one another at the time the boulder mass moved. Many of the big boulders are cracked and shattered, see Fig. 9. This is especially true of those that are well weathered. The fractures are contained entirely within individual boulders and do not pass from boulder to boulder or into the dolomitic cement. Some of them may represent joint planes that were present in the original granite mass; others, and these include most of the galena-filled cracks, seem more likely to have resulted from boulder impact or crushing.

The cement between the boulders is dense, fine-grained dolomite ranging in color from light gray green to dark gray. Much of it is sandy, especially near the Lamotte contact, and the dark gray parts are usually argillaceous.

Many of the large boulders are standing on end or in other odd positions such as the one shown in

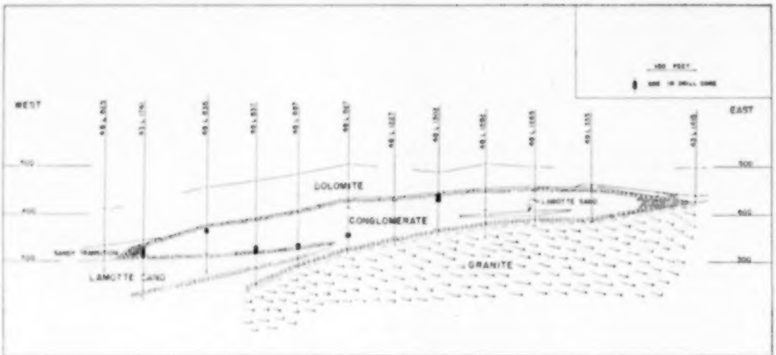

Fig. 5—Cross-section along line A-A'. See Fig. 4.

Fig. 10. Most of them are resting on other boulders or on top of the Lamotte sandstone. There is no sorting; large and small fragments occur scattered at random as individuals or in nests. Many of the small angular broken pieces are scattered through the dolomite and do not touch each other. Some of these are shown in Fig. 11. These fragments seem to have fallen into a calcareous ooze and been buried. The heavier large boulders would tend to sink until they encountered a more substantial base; hence they are not found in the suspended position of the small fragments.

A sandy ooze, which would have become the sandy transition beds in the Bonneterre, apparently overlay the Lamotte at the time (or times) that the boulders shifted. This ooze was disturbed until it lost any semblance of bedding. Some of it was stiff enough to support small granite fragments while letting the big boulders sink in; undoubtedly much of it was stirred up by the grinding force of the rock slide (or roll) only to settle back around the clastic fragments, cementing them together. The filling in process was very complete, and unfilled open spaces are almost nonexistent.

The degree of alteration of the fragments varies widely. Many pieces, small and large, are fresh red granite, while some of the boulders over 1 ft in diam are completely bleached to a dull gray. Quite a fair percentage have fresh interiors and an outer margin of alteration. Significantly the angular fragments, which have been ascribed to impact shattering, exhibit no zoning, being either entirely unaltered or else uniformly altered throughout. It is believed that the greatest share if not all of the alteration in the granite is caused by weathering, much of it probably ante-dating the movement of the boulders to their present location. Whether there is any ore solution alteration such as sericitization is yet to be determined.

Ore Occurrence

In the mine face the galena distribution is erratic and pockety, but the overall grade of the orebody is fairly uniform. The greatest share of the galena occurs in disseminated grains in the cementing dolomite. Crystals up to ½ in. in size may be found, but most of them are smaller. A lesser but still important share of the lead occupies the cracks within the boulders, see Fig. 12. These veinlets range up to ½ in. in width, but thin galena films less than 1 mm wide are more common. The cracks are tensional in character with sharp angular offsets and parallel walls. Quite frequently several parallel cracks lie close to each other as in Fig. 9. The crack-filling veinlets are found in altered boulders but do not occur in fresh red granite.

As shown in the cross-sections, Figs. 5 and 6, the largest continuous ore zone lies just above the top of the Lamotte in the lower part of the upper boulder bed and west of the "sand pinch-out." In some places the underlying top few feet of the Lamotte also contains pay ore. Other smaller, less continuous ore zones are found higher in the upper boulder bed, and some holes have shown considerable thicknesses of low grade mineralization between pay runs. The lower basal conglomerate occasionally is of minable grade. The overlying middle and upper Bonneterre beds above the boulder ore contain only traces of lead.

Galena is the only ore mineral, although traces of sphalerite have been noted. Marcasite and calcite constitute the bulk of the gangue mineralization.

Conglomerate Origin

No theory as to the method of accumulation of these huge masses of large rounded boulders can be supported with complete assurance until more has been seen underground, but some of the data now available have an important bearing on the problem. Two theories have been suggested, both of which explain certain features of the occurrence. One is that the boulder piles are simply talus accumulations on the upper slopes of the knobs which were caused to slide or roll down hill in early Bonneterre time. The original impulse that caused the piles to shift could have been faulting or wave action in the sea, which was rising in level at this time. In any event the theory proposes that the piles gravitated down the granite slopes and out onto the top of the Lamotte, stirring up the sandy Bonneterre ooze which, upon settling, filled in between the boulders and eventually cemented them into a solid mass. It seems likely that such movement would be concentrated in a few more or less violent actions.

There is evidence that faulting has occurred in the Hayden Creek area, which could have started the downhill shift of boulder accumulations on the upper slopes of the knobs, but the exact age of the fault movement is not known. Quite possibly it was recurrent. Fig. 7 shows an abnormally steep slope

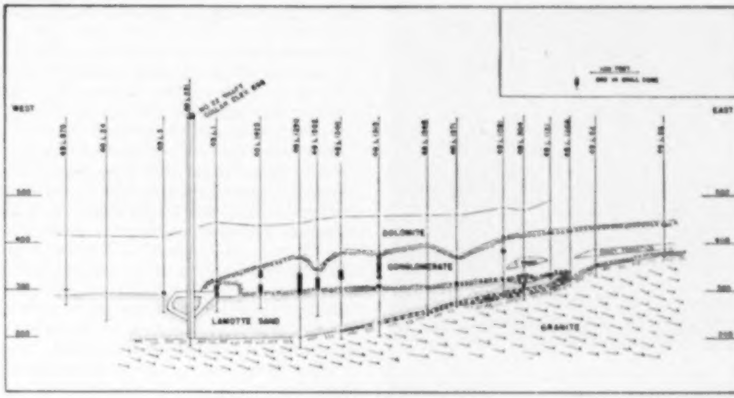

Fig. 6 — Cross-section along line B-B'. See Fig. 4.

Fig. 7 — Detail of the main South knob conglomerate body showing its relationship to the surface of the underlying Lamotte sand.

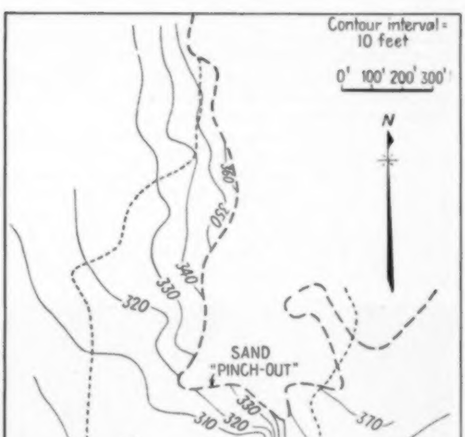

Fig. 8 — Thickness contour map of the main South knob conglomerate mass.

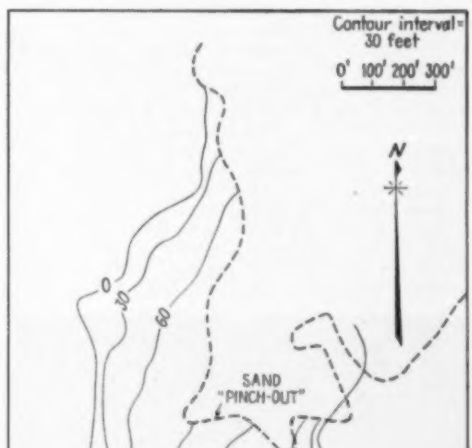

Fig. 9—Shattered granite boulder with galena filling the cracks.

on top of the Lamotte about 1000 ft east of No. 22 shaft. Part of this sharp drop in sand elevation to the southwest is the result of a zone of step faulting with northwest strike. If movement occurred along this fault zone at the critical time when a mass of boulders was in delicate equilibrium on the hillside, the mass might well have been set in motion to the southwest. Similarly, some minor early adjustment along the great Simms Mountain fault zone lying about ½ mile to the south might have triggered the action. The main Simms Mountain faulting, however, occurred long after the conglomerate was buried by younger sediments.

The talus theory explains very well the lack of size sorting and the impact shattering of the boulders. It is confronted with the serious problem, however, of why a talus pile should move southwest as it has from the South Knob when more direct downhill slopes were available to the west and southeast. Fig. 2, which shows the topography of the granite knobs at the time the boulders were formed, shows that there was only a rather small potential source area on the South Knob for boulders likely to move southwest. In the case of the North Knob, the main boulder tongue, being of much smaller volume than the South Knob tongue, is in better proportion to the size of the potential source area for its boulders. But, even here, the location of the main concentration is not a likely looking spot for gravity alone to have caused such a concentration of boulders. The talus theory offers no good explanation for the absence of similar boulder accumulations elsewhere around the knobs.

The smooth rounded exteriors of the large boulders might suggest that they had been rolled and abraded more than would be likely in talus migration downhill. It must be noted in this connection, however, that the outcropping granites in the Ozark region commonly round themselves quite remarkably by exfoliation, and it is probable that practically all of the rounding of the Hayden Creek boulders is the result of exfoliation and not a product of abrasion. This is true no matter what origin is advocated for the conglomerate.

What would seem to be a strong argument against the talus theory obtains from the calculation of the volume of boulders in the upper boulder bed and comparison with the potential source areas up the granite slopes. In the main boulder body on the South Knob, taking the boulders lying west and south of the sand "pinch-out line," there are 56 million cu ft. This does not include the considerable volume of upper boulders which cannot be positively identified as such because of the absence of Lamotte sand higher up the slopes but which surely is present as indicated by boulder thicknesses in excess of 100 ft just east of the "pinch-out." A layer of boulders 20 ft thick over an area 1680 ft square is equivalent to 56 million cu ft. Reference to Fig. 2 would seem to indicate that it would be most unlikely that the southwest slope of the South Knob could produce this volume of boulders at one time. Even if all of the solid granite available to erosion in this area were reduced to boulders and if all of the boulders formed moved in the right direction, i.e., southwest, it would be necessary for 15 to 20 ft of granite to be broken down to provide the boulders that are piled up in the No. 22 mine area. The time available for the accumulation in their present position is that period between the end of Lamotte time and the deposition of the non-sandy middle Bonneterre. Since this interval is quite short, there is serious question whether adequate time was available for weathering to produce from this small area

Fig. 10—Underground view showing the random orientation of large and small boulders.

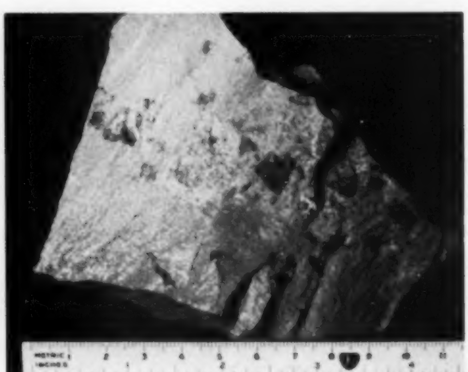

Fig. 11—Isolated angular granite fragments (dark gray) scattered through a matrix of dense sandy dolomite.

the boulders found in this one large conglomerate body.

The drillhole intersections of several feet of sandy dolomite within the conglomerate deserve some mention in this discussion of the history of the boulder mass. In Fig. 6, these intersections are interpreted as beds of sandy dolomite because they were logged as such when the holes were drilled. These holes did cut sandy dolomite at such depths that, on the section, it appears that there is a layer of this material in the midst of the upper boulder bed. If this is correct, it has significance because it indicates that there were at least two periods of boulder accumulation separated by an interval when more normal Bonneterre sedimentation occurred. Furthermore, it is shown that the deposition of the uppermost conglomerate was not so violent as to disturb a limey layer less than 10 ft thick and hence there probably could not have been talus migration with its attendant grinding.

Before any final conclusion is drawn based on this evidence, however, consideration should be given to another possible interpretation of these sandy dolomite core intersections. This conglomerate body consists of boulders up to 8 or 10 ft in diam cemented by sandy dolomite. The boulders are not tightly packed together, and in the mine, pockets of dolomitic cement of considerable size may be seen. Whether a drillhole cuts a granite boulder or the cementing dolomite at a given elevation depends on its location a few feet one way or the other. A drillhole passing between large boulders might show a section several feet long of sandy dolomite that entirely resembled normal basal Bonneterre rock and would be logged as such. Interpretation of the intersection as representing a bed of dolomite would be in error. Additional mine exposures will be necessary to ascertain whether the conglomerate is divisible into units separated by dolomite beds or whether it is entirely the product of one period of boulder deposition.

The other theory of conglomerate formation is that the tonguelike masses are the result of ocean current action, that is, boulder spits piled up at the ends of islands in the sea. The motivating force would be long-shore currents that rolled the boulders southward until they slid into deeper water at the ends of the knobs and came to rest. Such accumulation would probably have been more gradual than that advocated in the talus theory.

The location of the two main boulder piles strongly favors the ocean current hypothesis. Currents moving southward along the shores of the granite islands might be expected to carry along the rounded boulders and to build up a leeward accumulation. The huge size of some of the boulders and the fact that many of them are by no means spherical, though rounded, indicate that a strong current would be required.

If such a concentrated current was responsible for these boulder spits, it seems strange that it should have deposited its load without regard to size. The lack of sorting is very striking. The frontal slope of the piles is very steep and sharply defined and there is no trailing off of finer sized material. This is perhaps the strongest argument against the ocean current hypothesis, for even if the long shore current were related to a short, violent storm, some size sorting would be expected.

The proposal of a southward moving current is contrary to independent evidence in the Lead Belt

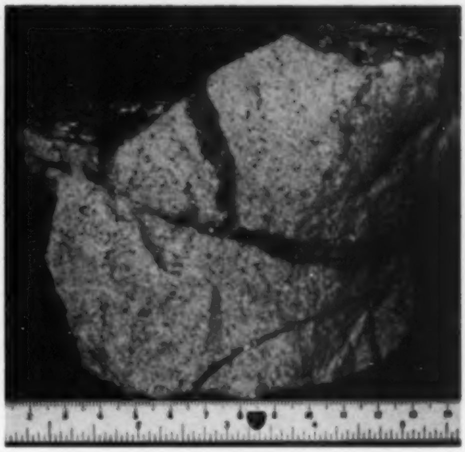

Fig. 12—Galena occupying tensional shatter-cracks through a granite boulder.

that the major currents of the area at this time were moving northeastward. The distribution of the sandy dolomite in the lower Bonneterre in long northeast-trending bars or ridges and their relations to the pre-Cambrian knobs are strong evidence for this belief.† The presence of Lamotte sand at higher elevations on the northeast sides of the knobs here and elsewhere in the Lead Belt is evidence that northeastward-moving currents also prevailed in Lamotte time. Thus the postulated southward current in the Hayden Creek area at this same time would have been purely local in scope.

The shattering observed in many of the weathered boulders is evidence of violent emplacement, and there is good reason to believe that once most of these boulders fell into their present positions and were shattered by impact they did not move again. Had they been shifted further, they would have fallen to pieces. Some of them did fall apart and gave rise to the smaller angular fragments, but most of them apparently did not. These observations support the belief that the piles were emplaced by short violent movement rather than gradual long-continued growth. The erratic, on-end positions of some of the large, rudely tabular boulders also suggests a lack of opportunity for shifting about and settling into the most stable position.

† Geological Staff, St. Joseph Lead Co. Unpublished Guidebook, AIME Annual Meeting Field Trip, February, 1951.

Summary

Prospecting in southeast Missouri in recent years has discovered several million tons of lead ore in an environment quite different from the usual southeast Missouri situation. The host rocks are masses of granite boulders which, in large part, overlie the Lamotte sandstone around buried pre-Cambrian granite knobs. Galena replaces the dolomitic cement and fills cracks in shattered boulders. The conglomerate bodies may represent talus accumulations or may have been caused by shore current action when the knobs were islands in the Cambrian Sea.

Acknowledgment

The author is indebted to the St. Joseph Lead Co. for permission to publish this report.

The Third Theory Of Comminution

by Fred C. Bond

MOST investigators are aware of the present unsatisfactory state of information concerning the fundamentals of crushing and grinding. Considerable scattered empirical data exist, which are useful for predicting machine performance and give acceptable accuracy when the installations and materials compared are quite similar. However, there is no widely accepted unifying principle or theory that can explain satisfactorily the actual energy input necessary in commercial installations, or can greatly extend the range of empirical comparisons.

Two mutually contradictory theories have long existed in the literature, the Rittinger and Kick. They were derived from different viewpoints and logically lead to different results.

The Rittinger theory is the older and more widely accepted.[1] In its first form, as stated by P. R. Rittinger, it postulates that the useful work done in crushing and grinding is directly proportional to the new surface area produced and hence inversely proportional to the product diameter. In its second form it has been amplified and enlarged to include the concept of surface energy; in this form it was precisely stated by A. M. Gaudin[2] as follows: "The efficiency of a comminution operation is the ratio of the surface energy produced to the kinetic energy expended." According to the theory in its second form, measurements of the surface areas of the feed and product and determinations of the surface energy per unit of new surface area produced give the useful work accomplished. Computations using the best values of surface energy obtainable indicate that perhaps 99 pct of the work input in crushing and grinding is wasted. However, no method of comminution has yet been devised which results in a reasonably high mechanical efficiency under this definition. Laboratory tests have been reported[3] that support the theory in its first form by indicating that the new surface produced in different grinds is proportional to the work input. However, most of these tests employ an unnatural feed consisting either of screened particles of one sieve size or a scalped feed which has had the fines removed. In these cases the proportion of work done on the finer product particles is greatly increased and distorted beyond that to be expected with a normal feed containing the natural fines. Tests on pure crystallized quartz are likely to be misleading, since it does not follow the regular breakage pattern of most materials but is relatively harder to grind at the finer sizes, as will be shown later. This theory appears to be indefensible mathematically, since work is the product of force multiplied by distance, and the distance factor (particle deformation before breakage) is ignored.

The Kick theory[4] is based primarily upon the stress-strain diagram of cubes under compression, or the deformation factor. It states that the work required is proportional to the reduction in volume of the particles concerned. Where F represents the diameter of the feed particles and P is the diameter of the product particles, the reduction ratio Rr is F/P, and according to Kick the work input required for reduction to different sizes is proportional to log Rr/log 2.[5] The Kick theory is mathematically more tenable than the Rittinger when cubes under compression are considered, but it obviously fails to assign a sufficient proportion of the total work in reduction to the production of fine particles.

According to the Rittinger theory as demonstrated by the theoretical breakage of cubes the new surface produced, and consequently the useful work input, is proportional to Rr-1.[5] If a given reduction takes place in two or more stages, the overall reduction ratio is the product of the Rr values for each stage, and the sum of the work accomplished in all stages is proportional to the sum of each Rr-1 value multiplied by the relative surface area before each reduction stage.

It appears that neither the Rittinger theory, which is concerned only with surface, nor the Kick theory, which is concerned only with volume, can be completely correct. Crushing and grinding are concerned both with surface and volume; the absorption of evenly applied stresses is proportional to the volume concerned, but breakage starts with a crack tip, usually on the surface, and the concentration of stresses on the surface motivates the formation of the crack tips.

The evaluation of grinding results in terms of surface tons per kw-hr, based upon screen analysis, involves an assumption of the surface area of the subsieve product, which may cause important errors. The evaluation in terms of kw-hr per net ton of −200 mesh produced often leads to erroneous results when grinds of appreciably different fineness are compared, since the amount of −200 mesh material produced varies with the size distribution characteristics of the feed.

This paper is concerned primarily with the development, proof, and application of a new Third Theory, which should eliminate the objections to the two old theories and serve as a practical unifying principle for comminution in all size ranges. Both of the old theories have been remarkably barren of practical results when applied to actual crushing and grinding installations. The need for a new satisfactory theory is more acute than those not directly concerned with crushing and grinding calculations can realize.

In developing a new theory it is first necessary to re-examine critically the assumptions underlying

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the old theories, which may have been accepted previously without due consideration.

The first of these assumptions to be questioned is concerned with the neglect of the work previously done on the feed particles. Any subdivided material represents work input, and the total work input to a material should be the primary consideration, not the work input increment below a certain feed size.

The second assumption which should be questioned is the derivation of both theories from the consideration of cubes under compression. Compressive stress applied evenly to parallel plane surfaces of cubes constitutes an almost nonexistent case in actual crushing and grinding, and applications of the theory derived from the theoretical breakage of the cubes into a certain number of smaller cubes are usually misleading. In actual operation the stresses are applied at protruding points of irregularly shaped bodies, these points crumbling until sufficient area is developed to break the particles under nonuniform stress. Theoretical consideration of the breakage of spheres under compression should be more applicable than that relating to cubes.

The strain energy theory[2] also employs the first and second assumptions and to this extent is open to the same objections as the Rittinger and Kick theories.

The third assumption is that useful work input is equivalent to energy content, or that the energy of a material after breakage is increased by the amount of useful work applied during breakage, even though this energy increment is not recoverable by any known means. This assumption is inherent in the second form of the Rittinger theory and underlies much of the current thinking on comminution. It postulates that the bulk of the energy input required in all known methods of breakage, which is released as heat, represents wasted work input.

These three assumptions were avoided in the derivation of the new Third Theory to be presented here. However, assumptions cannot be completely eliminated from the development of any theory.

The requirements of a successful theory should be considered:

1—It should give consistent results when applied to all comminution operations over all size reduction ranges, for all materials, in all types of machines. The different breakage characteristics of different materials over different size ranges, in different machines with varying efficiencies in each size range, should be properly evaluated by the new theory. After sufficient study under the correct theory, it should be possible to develop factors for the correction of each of the above differences and unerringly select the operation with the highest mechanical efficiency.

2—It should be supported by a large mass of operating data on a wide variety of materials correlated with laboratory tests on the same materials.

3—The correlation should be made by a work index, representing the total work input to a definite product size, which should be obtainable from any laboratory or commercial operation when the energy input and size analyses of feed and product are known.

4—It should be possible to calculate the useful energy input required for any size reduction from the work index or from any similar reduction in a different size range, when allowance is made for possible different breakage characteristics and machine efficiencies at different sizes.

Nomenclature

C	= Impact crushing strength in foot-pounds per inch.[10]
D	= Diameter of cubes or spheres.
F	= Feed size = 80 pct of feed passes in inches or microns.
Fc	= Cutoff size of scalped feed.
Fe	= Equivalent feed size of scalped feed.
Fp	= Percentage of feed passing product size P.
G_{bp}	= Ball mill grindability at product size.[10]
G_{rp}	= Rod mill grindability at product size.[10]
L	= Effective crack length in centimeters per short ton.
m	= Slope of percentage passing line on log-log plot. For normal material $m = 1/\sqrt{2}$.
P	= Product size = size 80 pct of product passes inches or microns.
P_1	= Grindability test sieve opening in microns.
Rr	= Reduction ratio = F/P.
S	= Specific gravity.
W	= Work input in kw-hr per short ton to any comminution operation.
Wi	= Work index = Wt for P = 100 microns.
Wp	= Total work input in kw-hr per ton represented by a product size P.
Wu	= Useful work input.
Wu/W	= Mechanical efficiency.
Y	= Percentage passing any diam X.
Yc	= Percentage passing cutoff size Fc of normal feed with 80 pct—size F.

5—It should have a theoretical basis confirmed by previously determined empirical relationships.

6—The absolute mechanical efficiency should be obtainable.

Definition of Terms

Work input, W, is the work or energy input in kw-hr per short ton to a machine reducing material from a definite feed size to a definite product size.

Total work input, Wt, represents the total work or energy input in kw-hr per ton applied to obtain a size reduction from a feed of theoretically infinite particle size to a definite product size. The total work input is the sum of the work input in reduction of the feed to the product and all work previously put into the feed.

Useful work input, Wu, represents that portion of the work input which causes breakage and eliminates that portion of the work input consumed in machine friction, wear of machine parts, contacts not including materials to be reduced, and contacts resulting in stresses below the critical breaking stress. It represents the theoretical minimum work input necessary at 100 pct efficiency. Its theoretical value has not been derived in the present paper. Mechanical efficiency is the ratio of useful work to work input, or Wu/W; it does not necessarily correspond to the thermodynamic efficiency, since comminution is not necessarily reducible to an adiabatic process, although it may have an adiabatic component.

The feed size, F, is the diameter in microns or inches of the square hole that will pass 80 pct of the feed and is found from the plotted screen analysis.

The product size, P, is the size 80 pct of the product passes and is found from the plotted screen analysis.

The reduction ratio, Rr, equals F/P.

The work index, Wi, is the calculated kw-hr per ton applied in reducing material of infinite particle size to 80 pct passing 100 microns, equivalent to approximately 65 pct passing 200 mesh. It establishes the relative reduction resistance of a material in the size range tested and the relative mechanical efficiencies of different machines and different processes. It can be found from any commercial or laboratory operation where the work input and size distribution of the feed and product are known.

The following definition of the comminution process is proposed in agreement with the concepts of the Third Theory.

The comminution process is a procedure in which mechanical kinetic energy is transformed into heat through internal and external friction, under conditions such that critical strains are exceeded and material is broken.

The mechanical efficiency is the ratio of the amount of breakage to the amount of energy transformed, or to the total energy input.

Derivation of the Third Theory

Experience has shown that the average slope, m, of the size distribution percent passing line of crushed and ground products, on a log-log plot of the percent passing against the sieve opening, is very close to $1/\sqrt{2}$, or 0.7071. The slopes vary somewhat with the nature and history of the material, but the average indicates that the normal slope of a homogeneous material that has been reduced by a nonselective comminution action is $1/\sqrt{2}$.

The normal size distribution line follows this equation:[5,6]

$$Y = 80 \left(\frac{X}{P}\right)^{1/\sqrt{2}} \qquad [1]$$

where Y is the percent weight passing any diam X, and P is the diameter 80 pct passes. The percent passing varies as the particle size X to the power m, which equals $1/\sqrt{2}$.

The surface areas of a unit volume of material in particles of similar shape of diam X vary as $1/X$. With normal materials the percent weight in each size fraction in the $\sqrt{2}$ sieve scale series varies as $1/\sqrt{2}$. Therefore, the surface area of each fraction varies as

$$X^{1/\sqrt{2}}/X \text{ or as } 1/X^{0.293}$$

If the slope m were unity, the surface area of each size fraction would be the same or would vary as $1/X^0$, while if the slope were ½, the surface areas would vary as

$$1/X^{1/2}$$

According to Rittinger the energy input required to break a cube of diam D varies as D^2, while according to Kick it varies as D^3. The strain energy absorbed by a cube under compression varies as its volume, or as D^3. However, with the formation of the first crack tip the strain energy effectively flows to the surface, which varies as D^2. When irregularly shaped pieces are broken, the strain energy is not evenly distributed throughout the rock, and the first crack tip forms and starts the energy flow, which develops a breakage pattern, when the proportionate energy absorbed is intermediate between D^2 and D^3. The average value is $D^{5/2}$.

The theoretical energy required to break spheres of similar material of diam D appears to be proportional to $D^{5/2}$.

Both surface and volume factors affect the breakage of stone, and when these effects are equal the energy to break should be proportional to $D^{5/2}$, which is midway between the Rittinger and the Kick assumptions.

The number of particles of similar shape in a unit volume varies as $1/D^3$, so that the energy input required to break a unit volume or unit weight should be proportional to $D^{5/2}/D^3$, or $1/\sqrt{D}$. This is the basis of the Third Theory.

Statement of the Third Theory

The Third Theory can be stated concisely as follows:

The total work useful in breakage which has been applied to a stated weight of homogeneous broken material is inversely proportional to the square root of the diameter of the product particles.

Where K represents a proportionality constant and P is the product diameter

$$Wt = K/\sqrt{P} \qquad [2]$$

Eq 2 is the fundamental statement of the Third Theory.

According to the Third Theory the work input W in breakage from a feed size F of one unit to a product size P of ¼ unit with a reduction ratio Rr of 4:1 equals the previous work input to the feed, or $Wt = 2W$.

For any values of F and P, where W is the kw-hr per ton required to break from F microns to P microns, the total work input, Wt, is proportional to $1/\sqrt{P}$, and W is proportional to $1/\sqrt{P} - 1/\sqrt{F}$.

$$\frac{Wt}{1/\sqrt{P}} = \frac{W}{1/\sqrt{P} - 1/\sqrt{F}} = \frac{W}{\dfrac{\sqrt{F} - \sqrt{P}}{\sqrt{F}\ \sqrt{P}}}$$

$$Wt = W\left(\frac{\sqrt{F}}{\sqrt{F} - \sqrt{P}}\right) \qquad [3]$$

and

$$W = Wt\left(\frac{\sqrt{F} - \sqrt{P}}{\sqrt{F}}\right) \qquad [4]$$

The term $\sqrt{F}/(\sqrt{F} - \sqrt{P})$ equals $\sqrt{Rr}/(\sqrt{Rr} - 1)$.

The work index Wi is the kw-hr per ton required to break from infinite size to $P = 100$ microns, and

$$Wi = W\left(\frac{\sqrt{F}}{\sqrt{F} - \sqrt{P}}\right)\sqrt{\frac{P}{100}} \qquad [5]$$

Eq 5 is not empirical but is derived directly from the previous statement of the Third Theory.

When the work index is known, the energy input W required to break at the same efficiency from any feed size F to any product size P, in microns, is found from

$$W = Wi\left(\frac{\sqrt{F} - \sqrt{P}}{\sqrt{F}}\right)\sqrt{\frac{100}{P}} \qquad [6]$$

When F and P are in inches the number 100 in eqs 5 and 6 should be replaced with 0.003937, and $\sqrt{0.003937} = 0.0627$.

The ratio of the work inputs required for any two reductions, W_1 and W_2, is

$$W_1/W_2 = \frac{\sqrt{F_1} - \sqrt{P_1}}{\sqrt{F_1 P_1}} \Big/ \frac{\sqrt{F_2} - \sqrt{P_2}}{\sqrt{F_2 P_2}} \qquad [7]$$

If breakage characteristics of a material remain constant over all size ranges, and the mechanical efficiency of all machines employed is the same, then the values of the work index calculated under all different conditions should be constant. The work index expresses primarily the resistance of a material to breakage. The variations reveal differences in the breakage characteristics at different sizes and differences in the efficiencies of varying machines and operations.

If a typical normal material requires 3 kw-hr per ton for reduction from a feed size F of 1600 microns to a product size P of 400 microns, then according to the Third Theory the previous energy input to the feed was 3 kw-hr per ton, and the work index Wi is 12.

The practical test of the correctness of the Third Theory, as well as the value of the work index, lies in the comparison of a large number of plant and laboratory results. If these show that the work index under a wide range of sizes and operations remains substantially constant or varies in a predictable and explainable manner, then the Third Theory must constitute the fundamental law of comminution. No such correspondence has ever been demonstrated for the other two theories.

Confirmation by Crushing Quartz

Quartz crystals were broken under slow compression and the new surface area produced was measured by gas adsorption.[7] When the reported average energy concentration at fracture was plotted by the present author on log-log paper against the new surface area produced per unit of energy input, the points obtained were somewhat scattered, but were bisected by straight line with a slope of $1/\sqrt{2}$. The equation for this line is

$$\text{sq cm/kg-cm} = 172/(\text{kg cm/g})^{1/\sqrt{2}}$$

or

$$\text{joules/sq m} = 59\ (\text{kw-hr/ton})^{1/\sqrt{2}} \qquad [8]$$

According to eq 8, at an energy input of 10 kw-hr per ton the input surface work of the quartz is 300 joules per sq m, while at 20 kw-hr per ton it is 492 joules per sq m. Tests have shown that about 300 joules per sq m are required on silicious ores,[8] but eq 8 shows that the input surface energy is not constant but increases as the $1/\sqrt{2}$ power of the energy input. This contradicts both forms of the Rittinger theory.

According to the Third Theory the input surface work should remain constant only when the slope m of the log-log plot of the percent passing size distribution line is 1/2. Since the normal slope is $1/\sqrt{2}$, the input surface work at this slope should vary as the energy input to the $\sqrt{2}/2$ or $1/\sqrt{2}$ power, showing that the Piret tests[7] confirm the Third Theory if it is assumed that the effective size distribution of the broken quartz follows the normal slope. The effective size distribution is affected by the use of large single crystals of feed and the abnormal breakage characteristics of pure crystallized quartz.

Confirmation by Breaking Spheres

A number of glass marbles of different diameters were broken by slow compression between parallel plane surfaces in the Allis-Chalmers Laboratory, the deformations measured, and the energy input values calculated.

The results were scattered, but the averages showed that the energy input required to break was approximately proportional to the diameter to the 5/2 power. The deformation at constant applied load appeared to be proportional to $1/D^{1/2}$.

Little information was found in technical literature on the energy required to break spheres, and the need for additional tests is very obvious. The data available at the present time tend to indicate the validity of the Third Theory as applied to spheres. However, the Third Theory may still apply to the reduction of broken rock even if it does not apply to the breakage of spheres.

Crack Length and Surface Area

The Third Theory has established that the total useful work done in crushing and grinding a stated weight of homogeneous material is inversely proportional to the square root of the diameter D of the product particles. It also postulates that the bulk of the necessary work input is utilized in deformation of the particles and released as heat through internal friction. Local deformation beyond the critical strain results in the formation of a crack tip, normally on the particle surface.[9] The formation of this crack tip is the immediate object of the work input. Once the crack tip is formed, the surrounding energy in the stressed rock immediately flows to the crack tip, which rapidly extends throughout the particle, splitting it and resulting in a break. The energy flow and stressed condition of the rock create additional crack tips, resulting in a breakage pattern. Little or no additional external energy need be applied to brittle materials to cause the break after the first crack tip is formed. The surface energy of the new surface formed may represent the useful work applied in splitting after the crack tip is formed, and may be supplied by the strain energy present in the deformed particle. The useful work input is essentially consumed in the formation of the crack tips and is directly proportional to the length of the crack tips formed.

When a single particle of diam D is broken, the average strain energy concentration per unit volume is proportional to the particle volume, or to D^3, while the surface area on which the crack tips form is proportional to D^2. The work necessary for breakage is proportional to the square root of $D^3 x D^2$, or to $D^{5/2}$.

The work necessary to break a unit weight or unit volume of rock is proportional to

$$D^{5/2}/D^3, \text{ or to } 1/\sqrt{D}$$

Since the length of the crack tips is directly proportional to the useful work input, it is necessary to define the crack length. It seems convenient to consider the crack length as that length which subtends an equal length of normal extension of the crack tips in each particle, since the particles concerned in crushing and grinding approach equal dimensions in the two normal breaking directions. If a break across a square cross-section, D units on a side, is considered, the crack length equals D, since the transverse length of the crack equals the distance of propagation of the crack. If a break across a cylindrical cross-section of diam D is considered, the crack length is $\sqrt{\pi D^2}/4$ or 0.886D.

The quantity L equals the total crack length formed in the reduction of a short ton of material.

Under this definition of L the new surface area formed by any reduction of a short ton equals $2L^2$.

Effective cracks are considered here to be crack tips that extend through the particle and split it without the external application of additional energy other than that already present in the stressed particle. Partial cracks, which do not completely split the less brittle materials, are not complete cracks under this definition, but they decrease the energy required for complete breakage in a subsequent reduction stage.

Since the total surface area formed in breaking a stated weight of rock to particles of diam D is inversely proportional to D, and the Third Theory useful work input is inversely proportional to $\sqrt{D}$, the useful work input is directly proportional to the square root of the surface area formed. The useful work input is directly proportional to $1/\sqrt{D}$ and also to the total crack length L, which equals the square root of one half the surface area formed by extension of the crack tips.

The total crack length, L, in the reduction of a short ton of rock, multiplied by the net energy input required to form a crack of unit length, gives the theoretical net energy input required, Wu, and this divided by the actual energy input required, W, gives the mechanical efficiency of any reduction process.

New techniques are required for laboratory measurements of the net energy input necessary to form a crack of unit length. The effective crack length equals the square root of one half the surface area formed; however, the measured surface area may include surface which did not result directly from the extension of the crack tips. The crack length is also related to the product diam D by a function which varies with the particle shape.

The unit cracking energy required in the commercial reduction of stone can be estimated as being in the order of 5 to 10 joules per cm. If an efficient grinding operation requires 300 joules per sq m of new surface area produced,[8] it requires 6 joules per cm of crack tips produced. Application of 10 kw-hr per ton, or 3.6×10^7 joules per ton, will result in the formation of a crack length L of 6×10^6 cm, or 60 km.

Variations in the slope m of the product size distribution line from the normal value of $1/\sqrt{2}$ greatly affect the new surface area produced, but have only the square root of this effect upon the energy input required.

The Rittinger theory is concerned with the measurement of surface areas, the Kick theory with the volumes of the product particles, and the Third Theory with the crack length formed. The Rittinger theory can be designated as the surface area theory, the Kick theory as the particle volume theory, and the Third Theory as the crack length theory.

Appreciation of the fact that the immediate objective of all crushing and grinding operations is the formation of crack tips will initiate a new stage of development in the theory of comminution, which will require years of study and controversy to complete. However, it promises to supply the actual explanation of the process which has hitherto eluded the Rittinger and Kick advocates.

Effect of Scalping Feed

Perhaps the most troublesome feature in the application of the Third Theory is the correction necessary for unnatural feeds which have had the fines removed by scalping. When the crusher feed has had part or all of the material passing the product size removed by screening, the energy input required per ton of feed is increased above that required for normal feed.

The necessary correction for a scalped feed when the work index is being calculated can be made as follows:

Let Fe represent the effective feed size, or the feed size of a normal feed, which would be equivalent in total work input per ton to the feed size F, which 80 pct of the scalped feed passes; Fe is larger than F. Let Fc represent the scalping cutoff size below which the fines in the feed have been effectively removed; Fc is known or estimated from the screen analysis of the scalped feed. Let Yc represent the percent passing the cutoff size Fc of a normal feed with 80 pct passing size F.

When the value of Fe has been found, the work index Wi for the scalped feed is calculated by substituting Fe in place of F in eq 5.

Solution Using Eq 1: In eq 1 Fc is substituted for X, F for P, and Yc for Y, and the equation is solved for Yc. Then in eq 1 $80\text{-}Yc/2$ is substituted for Y, F for X, and Fe for P, and the equation is transposed and solved for Fe. The transposed equation is

$$Fe = F \bigg/ \left(\frac{80 - Yc/2}{80}\right)^{\sqrt{2}}$$

and the combined equation for calculating Fe from Fc and F is

$$Fe = F \bigg/ \left(\frac{2 - (Fc/F)^{1/\sqrt{2}}}{2}\right)^{\sqrt{2}} \qquad [10]$$

Graphical Solution: On a log-log plot of the percent passing versus particle diameter a line with the normal slope $1/\sqrt{2}$ is drawn through the diam F, which 80 pct of the scalped feed passes to its intersection with the scalping cutoff size Fc, which determines Yc. A parallel line is drawn through the point F, $(80 - Yc/2)$; its intersection with the 80 pct passing line gives the value of Fe.

Specimen Calculations: 1—If the feed is normal and $F = 1600$, $P = 400$, and $W = 3$ kw-hr per ton, from eq 5 the work index $Wi = 12.00$.

However, if the feed is scalped at the product size, all particles smaller than 400 microns are removed, and F, P, and W have the above values, then $Fc = 400$, $Yc = 30.0$, $Fe = 2145$, and $Wi = 10.55$.

2—If the feed is all -10 $+14$ mesh, or all -1680 $+1190$ microns, $P = 400$, and $W = 3$, then $F = 1680 - \dfrac{(1680\text{-}1190)}{5} = 1582$, and $Fc = 1190$.

By substitution in eq 10 $Fe = 3330$, and from eq 5 $Wi = 9.18$.

Work Index Calculations

Some of the published data by which the Third Theory was checked against actual operations are contained in a former AIME publication.[30] The testing methods are described and test results listed on a large variety of ores and other materials. Empirical formulas have previously been computed by the author from actual plant operations by which these test results can be translated into kw-hr per ton required for size reduction at the average operating efficiency of the plants used for comparison. In these formulas the particle size is the size which 80 pct of the materials pass as found by graphs of the screen analyses. The grindability at the product

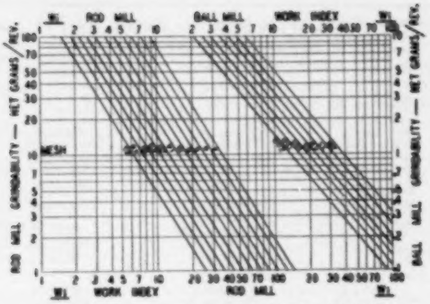

Fig. 1—Work indexes computed from rod and ball mill grindabilities.

size P is found by plotting the grindability for the various mesh sizes tested. If the grindability is available at only one size near the product size, a line is drawn through the plotted grindability parallel to the plotted percent passing size distribution line, to its intersection with the product size.

Impact Crushing Tests: It has been determined empirically from impact crushing tests,[20] and plant and pilot mill tests on the same materials, that the hp-hr per ton required to crush a scalped feed of specific gravity S and impact strength of C foot-pounds per inch of thickness, at a reduction ratio of 5:1, where P is the size in inches, which 80 pct of the product passes, equals $0.219C/SP^{[illegible]}$.

At a reduction ratio of 5:1 a normal feed has 25 pct passing the product size P, or $F_p = 25$.

If a feed with normal slope is used instead of a scalped feed, the exponent of P should increase from $1 - 1/\sqrt{2}$ to 1/2, according to the Third Theory, and the energy required to crush at any reduction ratio is

$$\frac{\text{hp-hr}}{\text{ton}} = \left(\frac{0.219C}{S\sqrt{P}}\right)\left(\frac{\sqrt{F}-\sqrt{P}}{\sqrt{F}}\right)\left(\frac{\sqrt{5}}{\sqrt{5}-\sqrt{1}}\right)$$

$$\left(\frac{80\text{-}25}{100}\right) = \left(\frac{0.218C}{S\sqrt{P}}\right)\left(\frac{\sqrt{F}-\sqrt{P}}{\sqrt{F}}\right) \qquad [11]$$

From eq 3 the total input energy required in kw-hr per ton is

$$Wt = \frac{0.1626C}{S\sqrt{P}}$$

and since 100 microns equals 0.003937 in., the work index from eq 5 is

$$Wi = 2.59C/S \qquad [12]$$

Eq 12 is used to calculate the work index from impact crushing tests. The work required to crush a normal feed from a given feed size to a given product size is calculated from the work index and eq 6, using 0.003937 in place of 100 when F and P are in inches.

Rod and Ball Mill Tests: Where G_{rp} is the rod mill grindability, and G_{bp} is the ball mill grindability[10] at the product size P, the kw-hr per net ton of material passing the product size (Wp) is found from the following empirical equations, which represent the average of a number of installations. The ball mill equation applies to a wet grinding overflow ball mill 7½ ft in diam inside the shell, operating in closed circuit with a classifier. The rod mill equation applies to a wet grinding overflow rod mill 6 ft in diam inside the shell, operating in open circuit.

$$Wp = 23/G_{rp}^{[illegible]} \qquad [13]$$

$$Wp = 20/G_{bp}^{[illegible]} \qquad [14]$$

The kw-hr per ton of new feed to both types of mills is found from the equation

$$W = 0.80\ Wp\left(\frac{\sqrt{F}-\sqrt{P}}{\sqrt{F}}\right) \qquad [15]$$

The total work input Wt at any size P is found by multiplying eq 15 by

$$\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}$$

and the work index is found by multiplying Wt by $\sqrt{P_1/100}$. Then

$$Wi = 0.80\ Wp\left(\frac{\sqrt{F}-\sqrt{P}}{\sqrt{F}}\right)\left(\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\right)\sqrt{\frac{P_1}{100}}$$

$$= 0.80\ Wp\sqrt{P_1/100} \qquad [16]$$

The values of $0.80\sqrt{P_1/100}$ were calculated for each mesh size at which tests have been made, where P_1 is the sieve opening in microns, and this factor multiplied by the Wp value obtained from the grindability tests by eqs 13 and 14 gives the work index for the test at that mesh.

The values of the work index factor, $0.80\sqrt{P_1/100}$, for each mesh size in both rod and ball mill tests are as follows:

Mesh	Wi Factor	Mesh	Wi Factor
3	6.54	28	1.942
4	5.52	35	1.639
6	4.637	48	1.378
8	3.902	65	1.158
10	3.280	100	0.9785
14	2.760	150	0.8190
20	2.318	200	0.6883

Work indexes can be found directly from rod and ball mill grindabilities by Fig. 1 or an enlargement thereof.

Proof of Third Theory

From Laboratory Tests: Since consistent data covering a wide variety of materials, size ranges, and machines are necessary to prove the correctness of a comminution theory, the work index has been calculated for reported materials on which more than one test has been made.[20] Comparison of these Wi values for different rod and ball mill grinds and for crushing tests shows the correspondence obtained; differences at different size ranges in these results, which are greater than the experimental error of about 5 pct in sampling and testing, are caused principally by differences in the ease of breaking at different sizes.

The general consistency of the work index values obtained for a large number of different materials and conditions by the Third Theory and by empirical equations, which fairly represent actual plant operating results, is regarded as conclusive proof of the correctness of the theory. Nothing approaching such consistency has ever been shown in support of the Rittinger or Kick theories, neither of which has been particularly helpful in actual crushing and grinding operations.

These tests are listed alphabetically in Table I; they include 559 tests on 144 different materials. The

Table I. Work Indexes Calculated from Allis-Chalmers Laboratory Tests, Work Index = Kw-Hr Per Ton from Infinite Feed Size to 80 Pct Passing 100 Microns, or About 65 Pct Passing 200 Mesh.

Key: BM Mesh - Work Index = Ball Mill Mesh - Work Index
RM Mesh - Work Index = Rod Mill Mesh - Work Index
IC S - Work Index = Impact Crushing Sp. Gr. - Work Index

Name	Test	Mill	Mesh—Wire Index	Mesh—Wire Index	Mesh—Wire Index
Ajo-New Cornelia	684	BM	28-18.27	35-16.40	48-14.95
	2081	BM	65-14.05	100-13.47	200-14.45
		IC	2.69-21.36		
Alan Wood Steel	931	BM	35-8.34	48-8.20	65-9.76
Alcoa-Pet. Coke	700	BM	35-105.7	48-73.3	100-76.4
Al-Ke-Me Fertilizer	252	BM		48-13.33	100-15.10
Alum. Co. of Canada	1080	BM		48-6.03	65-7.06
	2149	RM		48-8.21	65-7.80
		IC	2.87-11.49		
Alum. Ore Co.	1858	RM	10-10.74	14-9.52	
Anaconda 1	910	BM	28-11.46	35-11.90	48-12.06
	1226		65-11.05	100-11.73	150-12.36
			200-14.00		
Anaconda 2	1477	BM	28-12.65	35-12.20	48-10.97
			65-10.80	100-11.81	150-12.86
			200-12.04		
Bagdad, Ariz.	2623	BM	48-11.72	65-11.12	200-12.45
Basic Refractories	1033	BM	48-6.82	65-7.02	
	2643	BM	100-6.07	200-8.93	
Benguet Cons.	550	BM	28-15.00	35-16.06	48-14.39
			65-14.50		
Bernheim-Rand	477	BM	28-31.9	48-30.8	100-33.6
			200-22.0		
	2016	IC	2.93-23.40		
Black River Falls	1023	BM	100-8.58	150-9.61	
Bingham		BM	48-6.38	100-7.82	200-8.28
Bolidens Gruv.	2324	BM	200-8.89		
		RM	10-14.10	14-13.70	
		IC	3.81-9.32		
Bowring	288	BM	35-13.13	35-13.40	65-13.48
Broken Hill—Lead	2422	BM	100-11.25	150-11.87	
		RM	14-6.27		
Broken Hill—Zinc	2422	BM	100-12.33	150-13.90	
		RM	14-7.26		
Buffalo Ankerite	1639	BM	200-17.02		
		RM	14-21.90		
Bunge Corp.	1555	RM	14-17.89	48-16.55	
Butte Highlands	861	BM	100-7.64	200-8.38	
Can. Nephelene-Syenite	663	BM	28-9.52		
		RM	28-11.95		
Carlota	600	BM	100-14.03	200-13.26	
C&H Tailings 1	771	BM	35-27.4	100-22.6	200-20.1
2	2411	BM	100-16.84	200-17.21	
		IC	2.82-15.86		
Castle Dome—Miami	1042	BM	28-13.42	35-13.23	48-12.88
			65-13.02	100-12.12	150-12.45
			200-13.37		
Chino-Nevada Cons.	1060	BM	28-8.40	35-10.24	48-9.45
			65-10.28	100-10.34	150-10.81
			200-11.42		
Cleveland Cliffs	2123	RM	10-18.77	14-15.73	
		IC	3.3-13.42		
Climax Moly.	1560	BM	100-9.47		
		IC	2.62-13.95		
	2596	IC	2.67-12.24	2.65-10.25	
Cline Lake	745	BM	65-11.66	100-11.38	200-11.67
CMS—Golden Rose	864	BM	48-9.22	100-9-10	150-9.17
			200-10.30		
CMS—Kimberley 1	1682	RM	10-17.08	14-15.38	
2	2150	RM	10-13.07	14-11.14	
		IC	2.97-14.39		
Cobrecite	830	BM	35-6.56	65-4.79	
Cohart Refract. 1	809	RM	4-49.80	10-41.00	20-38.80
2			20-29.45		
Cons. Copper, Nev. 1	186	BM	48-6.96	65-8.51	
2	276	BM	48-11.24	65-12.57	
Cons. Feldspar	1103	BM	35-8.04		
		BM	28-12.81	35.10.06	
Cyprus Mines	432	BM	65-12.47	100-12.10	200-11.21
East Malartic	779	BM	28-9.73	48-9.36	65-9.04
			100-9.24	150-8.60	200-8.28
East Sullivan	1823	BM	100-7.44	200-8.81	
Massive		RM	14-11.85		
East Sullivan	1823	BM	100-13.90	200-14.10	
Normal		RM	14-18.50		
Elect. Met. Chrom.	1008	BM	65-60.3	100.60.1	
		RM	28-30.0		
Emsco Refractories	1178	BM	48-7.55	65-9.70	
Exolon-SiC	1052	RM	8-9.49	20-20.08	20-19.07
			20-15.20	28-13.79	
Federal Chem.	814	RM	8-8.70	28-9.56	
	533	RM	35-12.45		
Ford Motor	1035	BM	100-15.67	200-19.20	
Fresnillo	2675	BM	100-16.40		
		RM	14-17.65		
Getchell Mines	1241	BM	35-10.63	65-11.50	100-11.59
			200-11.50		
Golding-Keene	784	BM	200-11.37		
	1553	RM	14-11.84	20-10.65	
Granby Cons.	1566	BM	200-21.13		
Graphite Conc.	860	RM	14-21.60		
		BM	150-42.6	200-43.8	
Grootvlei—					
Banket	2054	BM	48-13.30	100-14.30	200-16.50
BM Feed	2054	BM	48-13.00	100-14.60	200-14.90
Shale	2054	BM	48-13.00	100-14.00	200-12.60
Quartzite	2054	BM	48-13.00	100-15.60	200-16.60
Hanna—Taconite	1397	BM	28-11.17		
	1022	RM	20-11.23	28-11.17	35-9.40
	2240	IC	4.24-9.57		
Homestake	1963	BM	28-12.68	48-11.30	65-10.74
			100-11.18	200-11.33	
		RM	10-15.40	28-11.76	
		IC	3.15-8.45		

Name	Test	Mill	Mesh—Wire Index	Mesh—Wire Index	Mesh—Wire Index
Industrial Silica	1204	RM	10-11.94	20-15.05	
Inland Steel	837	BM	48-6.27	100-8.58	150-8.73
	1023		150-9.61		
Int. Nickel	2128	BM	200-14.60		
		RM	14-20.63		
		IC	2.92-19.00		
Ipanema	352	BM	65-11.94	100-14.15	
H. J. Kaiser 1	677	BM	65-9.08	200-11.39	
2		BM	65-7.05	200-9.38	
Kerr Addison 1	799	BM	28-20.24	35-17.30	150-13.00
			200-12.21		
		RM	14-16.29		
2	1765	BM	28-14.30	35-13.43	150-11.37
			200-12.02		
Kelowna Exp.	861	BM	48-21.20	65-19.22	100-19.41
			150-18.35	200-17.55	
Kinetic Chem.	763	RM	14-14.06	48-11.02	
Lake Shore	1831	BM	200-16.73	200-16.55	
	1644	RM	6-21.40	8-19.82	10-18.80
			14-20.70	14-18.25	
LaLuz	832	BM	28-18.50	35-17.73	48-16.54
			65-16.11	100-15.62	
Lawrence Cement	972	BM	150-9.76	200-8.86	
Limestone Prods.	1700	RM	14-11.19	35-9.34	
Little Long Lac	570	BM	28-17.00	35-18.25	48-16.90
			65-16.80	100-17.40	150-15.34
			200-14.90		
Madsen Red Lake	626	BM	65-14.73	100-13.80	
Magma-San Manuel 1	2419	BM	48-13.63	65-13.10	100-13.21
			200-11.33		
		RM	14-14.23		
2	2563	BM	65-13.31	100-14.15	200-12.66
		IC	2.58-6.24	2.63-6.60	2.53-7.87
			2.59-10.72		
Malartic	586	BM	28-15.35	35-16.40	48-13.98
			65-13.05	100-12.69	150-12.88
			200-13.44		
	1392	RM	10-18.38		
	1853				
	1952	IC	2.71-22.00		
Maine Dev.	1679	BM	100-13.22		
		RM	20.15.20		
Marievale, S.Af.	2050	BM	48-14.30		
		RM	14-16.27		
McIntyre Porcupine	2036	BM	65-12.98	200-13.18	
		RM	6-17.96	14-14.78	
Miami Copper	1042	BM	28-14.35	48-12.46	100-11.94
			200-12.37		
Mines Dominiales	726	BM	65-10.24		
	1716	RM	35-8.93		
		IC	2.15-3.42		
Mines de Bor	249	BM	150-12.27	200-7.99	
Mineral Mining	782	BM	35-6.07		
		RM	35-11.14		
Minn. Mines	637	BM	65-11.30	150-12.63	
Moose Mountain	756	BM	65-11.50	150-10.65	200-11.62
		RM	8-20.44		
Montecatini	469	BM	48-12.15	65-11.36	
Morenci	913	BM	28-8.53	35-9.60	48-10.20
	1937		65-10.85	100-11.54	150-12.45
			200-12.80		
		RM	20-9.96	28-9.52	35-11.78
			48-11.70		
		IC	2.06-13.85		
Moran	914	BM	65-7.20		
		RM	20-21.85		
Nat. Lead-Tahawus	1017	BM	35-9.02	48-10.46	65-9.92
	1757		100-11.92	150-13.70	200-10.32
	1800	RM	14-13.38	20-11.60	28-11.28
	1941		65-13.96		
		IC	4.46-14.10		
Nevada Cons. McGill	188	BM	48-11.24	65-12.57	
	278				
New Jersey Zinc	1951	BM	48-15.53		
		RM	14-24.42		
N.W. Magnesite	1031	BM	65-8.61	100-9.46	
	1911	IC	3.00-9.15		
Noranda		BM	48-19.98	100-18.25	200-16.00
Oglebay Norton					
Taconite	1167	BM	28-10.24	35-10.56	48.10.00
			65-10.57	100-10.64	150-11.57
			200-12.06		
Ozark Ore Co.	1889	RM	8-21.60	10-19.70	14-18.36
	1979				
	2355	IC	3.40-17.62	2.88-18.50	
Pacific Coast					
Aggregates	324	BM	28-26.8	35-26.5	
Parcoy	567	BM	65-11.30	150-11.78	
Picacho	880	BM	48-11.18	65-11.06	65-13.70
	517	RM	8-9.65		
Pickands-Mather					
Taconite	1803	RM	10-24.62	14-21.13	20-19.71
		IC	2.92-26.93		
Pitts. Coal Co.	1861	RM	14-9.83	28-8.34	
Pitts. Met. FeSi	639	RM	28-6.72	35-7.38	
	1612	IC	6.53-9.95		
Pitts. Plate Glass 1	1018	BM	100-10.07	150-10.72	
2		BM	100-9.67	150-10.16	
Portland Gold		BM	28-22.20	48-18.30	100-16.14
			200-13.38		
Powell-Rouyn	949	BM	150-13.58	200-14.20	
Preston East Dome	604	BM	150-9.26	200-8.88	
Quartz-Crystallized		BM	28-11.16	48-12.05	100-13.72
			200-15.29		
Queenstown-Gold	2082	BM	65-15.21		
		RM	6-23.00		
Quincy	1036	BM	28-15.00	35-17.20	48-15.83
			65-15.91		
Rand-Springs Mines	504	BM	28-14.77	35-15.80	48-15.65
			65-14.85	100-15.44	150-15.06
			200-15.50		
Real del Monte	730	BM	28-16.14	35-15.48	48-14.74
		BM	100-14.65	200-18.58	

Name	Test	Mill	Mesh—Wire Index	Mesh—Wire Index	Mesh—Wire Index
Rep. Steel					
Chateaugay	868	BM	28-8.70	35-9.18	
Harmony	868	BM	28-8.70	35-9.18	
	822	RM	20-8.58	35-9.29	
	1377	IC	3.29-5.18	3.30-4.46	
Old Bed	868	BM	28-8.70	35-9.18	
	822	RM	20-6.90	35-9.50	
	1377	IC	3.29-5.18	3.30-4.46	
New Bed	822	RM	20-6.66	35-9.67	
Reserve Taconite					
(Different samples)	1748	RM	14-16.28	14-25.70	
	2476	RM	14-18.80	14-13.80	
	2475	RM	14-18.50	14-21.42	14-17.75
	2150	IC	3.50-11.10		
	1877	IC	3.76-14.86	3.75-15.00	3.73-19.28
	1748	IC	3.07-15.88		
	1456	IC	3.16-16.29		
	39R	IC	3.48-12.07		
Rochester-Plymouth	818	BM	100-11.86	200-12.36	
St. Joe. Lead					
(Different samples)	847	BM	48-6.76	65-13.47	200-12.37
	2632	BM	65-11.59	100-11.25	200-9.55
	1657	RM	14-15.49	14-9.17	
	2632	IC	2.90-15.04		
San Luis	730	BM	28-16.14	35-15.48	48-14.74
			100-14.65	200-18.58	
Santa Maria del Oro	574	BM	65-11.60	100-13.42	
Sherritt-Gordon	206	BM	48-11.27	65-12.14	100-12.98
	1970		200-11.15		
Silver Bell	1050	BM	48-9.30	65-9.92	200-12.23
South Am. Dev.	752	BM	48-8.67	65-9.29	100-9.28
			200-10.86		
Sullivan Mines	740	BM	28-4.31	35-5.72	
Sutton Steele & Steele	530	RM	20-10.67	35-11.30	
Sydvaranger	1921	RM	20-19.95		
		IC	3.36-20.72		
Tenn. Copper Co.					
Isabella	2287	BM	48-7.65	65-8.72	100-9.00
	1750	RM	14-10.90	14-10.45	
	2613	IC	2.74-8.80		
London	2287	BM	48-6.27	100-8.50	
	1750	RM	14-7.45	35-7.50	
	26.3	IC	2.81-5.93		
Tri-State Flint	440	BM	48-30.3	100-29.0	200-24.5
TVA-Fontana	1055	BM	48-9.43	100-9.29	
	1372	RM	14-10.10		
	1750	IC	2.65-9.42	2.60-15.64	
Union Potash					
Langbeinite	842	RM	14-6.35	35-8.35	48-10.60
Sylvanite	842	RM	35-8.35	48-11.00	
Utah Copper	1763	BM	28-12.64	35-10.67	48-10.44
			65-10.41	100-10.35	150-10.97
			200-11.49		
	938	RM	14-13.45	20-12.80	28-12.25
			35-11.45	48-11.00	
		IC	2.57-12.64		
Volunteer-Cement					
Clinker	828	RM	3-18.47	4-16.56	10-15.20
Waite-Amulet	1761	RM	14-19.14		
		IC	3.35-10.72		
Western Minerals	883	BM	48-3.55		
Tripoli		RM	48-9.51		
Western Mining					
Australia	2193	BM	100-10.20		
		RM	14-17.65		
		IC	2.91-20.10		
Weston Brooker	1864	RM	4-19.61	6-17.94	
	1719	IC	2.65-21.85		
White Pine (Copper Range)					
Shale	1000	BM	28-15.57	35-14.76	48-13.92
	2078		65-13.48	100-13.57	150-12.25
			200-13.14		
Sandstone 1	1000	BM	28-11.50	35-10.35	48-11.16
			100-13.40	150-13.59	200-13.36
2	2026	BM	28-6.52	35-9.50	48-10.71
			100-16.14	65-15.44	200-15.76
		RM	48-13.82		
Amygdaloid	1000	BM	28-24.4	35-23.0	48-21.2
			65-20.26	100-19.84	150-18.45
			200-17.70		
		BM		48-20.68	
Wright-Hargreaves 1	406	BM	28-30.5	48-23.0	100-19.95
			200-16.94		
2			48-17.09	100-17.40	200-16.80

Table II. Work Indexes Calculated from Plant Data in Taggart[11]

Machine Plant	Ref. Taggart, Sect., Page	KW-Hr/Ton, W	Feed, F	Product, P	Work Index		Fe
					Plant	Test	
Gyratory crusher							
Utah copper	4-28	0.0832	206,000	152,400	12.0	11.49	315,000
Chino	4-27	0.0643	(76,200)	76,200	7.5	10.13	130,000
Ajo 1	4-28	0.1400	183,000	152,400	20.0	21.36	290,000
(New Cornelia) 2	4-28	0.1316	206,000	177,800	20.9	21.36	330,000
Cone crusher							
Utah copper	4-50	0.166	26,700	18,000	6.8	11.49	40,000
Chino	4-50	0.219	26,700	22,100	11.8	10.13	42,000
Crushing rolls							
Utah copper	4-64	2.240	27,920	795	7.6	11.49	29,000
Ajo (New Cornelia)	4-67	0.910	11,690	2,540	8.0	21.36	14,000
McIntyre Porc.	4-65	0.530	97,500	34,000	20.3	17.96	126,000
McIntyre Porc.	4-67	0.633	12,800	7,300	14.9	17.96	18,000
C & Hecla	4-61	2.175	16,000	3,000	20.2	20.00	18,000

Table II (continued). Work Indexes Calculated from Plant Data in Taggart[11]

Machine Plant	Ref. Taggart, Sect., Page	KW-Hr/Ton, W	Feed, F	Product, P	Work Index Test	Work Index Plant	Fe
Rod mill							
Chino (Old)	5-43	1.830	1,500	377	11.18	10.13	
Tenn. Cop—London	5-42	2.91	13,400	365	6.65	7.45	
Fresnillo	5-43	7.41	19,800	670	23.35	17.65	
Morenci 1	5-42	1.418	2,600	665	7.40	9.98	
Morenci 2	5-42	0.917	2,850	1,150	8.50	9.98	
Ball mill, overflow							
Utah copper	5-57	1.67	1,510	650	12.36	11.49	
Chino	5-57	7.24	770	129	13.91	10.13	
Ajo (New Cornelia)	5-56	7.37	5,400	150	10.83	13.47	
White Pine (C.R.)	5-57	7.94	3,730	208	17.63	20.66	
Ball mill, grate							
Chino	5-82	3.88	1,150	200	13.26	10.13	
Wright-Hargreaves	5-84	8.60	5,300	208	19.31	20.25	
Ball mill, conical							
McIntyre Porc.	5-64	4.81	23,000	600	14.05	13.08	
Wright-Hargreaves	5-65	10.33	4,800	300	23.85	20.25	
Miami	5-64	1.787	15,000	1,710	11.15	12.78	
Miami—Sec.	5-66	9.55	2,190	140	15.12	12.78	
Miami—Sec.	5-66	4.66	2,300	340	13.94	12.78	
Buffalo Ankerite	5-64	5.06	8,300	700	18.85	19.46	
Sherritt-Gordon	5-67	2.985	4,200	750	14.15	11.89	
Anaconda	5-68	4.38	30,000	805	14.87	12.00	
Tube mill							
McIntyre Porc.	5-91	5.84	1,580	230	14.31	13.08	
Kelowna	5-91	16.00	1,700	67	16.38	17.55	
Kelowna	5-91	10.76	151	51	18.33	17.55	

Work Index	Utah	Chino	Ajo	McIntire Porcupine	Wright-Hargreaves
Gyratory	12.0	7.5	20.0 (26.9)		
Cone	6.8	11.8			
Rolls	7.6		8.0	14.9 (20.3)	
Rod mill		11.18			
BM—overflow	12.36	13.91	10.83		
Grate		13.26			19.31
Conical				14.05	23.85
Tube mill				14.31	
Test IC	12.64		21.36	(17.96)	
BM	11.49	10.13	13.47	13.08	20.25

work input required for any feed and product size of any material listed is found from the work index nearest that product size and eq 6.

It is noted that the work index of pure crystallized quartz increases regularly from 28 mesh to 200 mesh, with a slope of −0.15 on a log-log plot against the product size, indicating that a general theory of comminution should not be based on the behavior of this material unless corrections are applied.

From Direct Plant Data: Work indexes were calculated from published plant operating data for comparison with each other and with those calculated from laboratory test results and are listed in Table II. All the published operating data are from Taggart's *Handbook of Mineral Dressing,*[11] with references to section and page. The examples were chosen largely at random, for the purpose of enabling those interested to become familiar with the calculation method, and have no particular significance in themselves other than to demonstrate the utility of the Third Theory in comparing relative efficiencies over the complete comminution size range. No such consistent comparison is possible under the Rittinger or Kick theories.

Much of the published data on crushing is not sufficiently complete for the calculation in Table II. To obtain uniformity in the crusher calculations, the cutoff size Yc was assumed to equal one half the product size P, which was taken as the open side crusher setting. The feed size F was considered equal to 15 pct of the crusher gape. These arbitrary approximations were necessary in the absence of exact size analyses, and the work indexes calculated therefrom are of value chiefly in illustrating the correction methods for scalped feeds.

The work indexes were calculated by eq 5 from the kw-hr per ton, feed size, and product size derived from the published data. In cases where the feed is marked Fe the equation is modified, as previously described. Similar corrections for closed circuit grinding installations are usually small and were not made because of incomplete data. Comparisons of the relative efficiencies of different plants can be made by dividing the plant work index by the laboratory work index. Additional comparisons can be readily made by anyone interested in a particular ore.

Work Index Use

The work index Wi can be calculated from laboratory tests by the methods shown and can be calculated for any plant operation by eq 5 from the feed and product sizes and the work input in kw-hr per ton. The kw-hr per ton required for any size reduction can be calculated from the work index by eq 6. The correction described should be applied when the feed is scalped.

When laboratory or plant results show an appreciable and consistent difference in the work index at different product sizes, indicating a difference in the breakage characteristics, the work index at the proper size should be used. Higher work indexes for scalped feeds may indicate concentration of a hard fraction in the scalped feed, which can be confirmed by additional laboratory tests made upon it.

Judgment must be used in each case regarding the effects on the relative efficiency of such variables as: different breakage characteristics at varying sizes and natural grain sizes, segregation of harder fractions, different machines, oversize feed, ball and rod sizes, circuit change, and differences in circulating loads. After the work indexes are obtained for a variety of conditions, correction factors for each variable can be developed, and the range of the judgment factor required at present will be decreased still further.

The relative efficiencies at different plants can be compared by dividing their work indexes by those obtained from standard laboratory tests on their ores, and the most efficient operation can then be selected. The work index of a material is the most valuable single item of information regarding its crushing and grinding that can be obtained at the present time.

The comparisons made to date show much less variation in the relative mechanical efficiencies of various crushing and grinding installations than had been expected. This agreement in relative efficiencies between different machines and circuits indicates that the absolute mechanical efficiency of existing machines, as yet undetermined, is comparatively high and is certainly far above the 1 pct or so postulated by the second form of the Rittinger theory. The determination of the absolute mechanical efficiency of existing machines is important in evaluating the possible savings to be made by new machines and methods.

Summary and Conclusions

A completely new theory of comminution has been derived by the author, stating that the total work required for crushing and grinding varies inversely as the square root of the product size. When the work input, feed size, and product size of one reduction are known, the theory permits rapid calculation of the work required for any size reduction and permits direct comparisons of efficiency. The relative efficiencies of different operations and circuits involving different feed and product sizes can be compared directly, and after laboratory tests have been made the relative efficiencies of plants reducing various materials can be compared.

The Rittinger and Kick theories are not correct and cannot be applied in a practical manner to ordinary crushing and grinding over a large size range.

Crushing and grinding do not follow the laws of the theoretical breakage of cubes.

This new Third Theory states that the total work useful in breakage that has been applied to a stated weight of homogeneous broken materials is inversely proportional to the square root of the diameter of the product particles, directly proportional to the length of the crack tips formed, and directly proportional to the square root of the surface area formed.

The immediate objective of crushing and grinding is the formation of cracks, which are then propagated through the deformed particles by the flow of resident energy to the crack tips. The essential work input is that necessary to deform the particles beyond their critical strain and thus to form crack tips. The total useful work input per ton of material reduced from theoretically infinite particle size is directly proportional to the effective length of the crack tips formed, which is directly proportional to the square root of the surface area formed and inversely proportional to the square root of the diameter of the product particles.

When F is the diameter that 80 pct of the feed passes, P is the diameter that 80 pct of the product passes and W is the work input in kw-hr per ton; then Wt, the work input required to reduce from an infinite feed size, is

$$Wt = W\left(\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\right)$$

When F and P are in microns and Wi is the work index, or total kw-hr per ton required to reduce from infinite size to 80 pct passing 100 microns, or to approximately 65 pct passing 200 mesh,

$$Wi = W\left(\frac{\sqrt{F}}{\sqrt{F}-\sqrt{P}}\right)\sqrt{\frac{P}{100}}$$

If F and P are in inches the number 100 should be replaced by 0.003937.

If the feed has been scalped, the equivalent feed size of a normal feed should be used.

The work index Wi can be calculated from laboratory tests, or from any plant operation where W, F, and P are known, by the above equation. When the work index is known, the work input required for any size reduction can be calculated by transposing the equation.

The work index should be constant within the experimental error over all size ranges for a homogeneous material reduced at the same relative mechanical efficiency. The work index is the key figure in any crushing or grinding problem.

The work index has been calculated and compared for a large number of laboratory impact crushing, ball mill grinding, and rod mill grinding tests and from published data for large plant installations. The values are consistent and reasonable over all size ranges. They permit comparisons of relative efficiencies.

Judgment must be used in applying the work index to specific problems to allow for variables such as: different breakage characteristics at different size ranges, concentration of harder fractions when part of the plant feed has been removed, circulating loads, machine speeds and sizes, oversize in feed, and pulp dilution. After work indexes for the different variables have been obtained correction factors may be determined.

The Third Theory and its work index permit closer predictions and more accurate comparisons of all crushing and grinding installations than have heretofore been possible and should result in more efficient operations.

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Deleading Zinc Concentrate

At the

Parral and Santa Barbara Mills

by C. L. Boeke and G. G. Gunther

THE zinc deleading processes at the Parral and Santa Barbara mills are described separately to provide a basis for comparison. Although the two procedures are fundamentally alike, there are some differences in application to meet specific conditions.

The Asarco mill at Parral in the State of Chihuahua is a flotation plant treating 1450 metric tons per day of complex sulphide copper-lead-zinc ores. In general, the procedure used for selective separation of the minerals into a copper, a lead, and a zinc concentrate follows the usual practice.

First, a bulk copper-lead concentrate is floated and then separated into a final copper and a final lead concentrate by the sulphurous acid process. A small tonnage of high grade gravity lead concentrate is tabled from the bulk copper-lead cleaner tailing before it is reground so as to avoid undesirable overgrinding and sliming of this portion of the lead mineral. Next, a zinc concentrate is floated from the lead rougher tailing and, as described later, is separated into a final lead product and a final zinc concentrate. Finally, the zinc rougher tailing receives a strong scavenger float to remove all recoverable middling values for regrinding and return to the flotation circuits. The slime is separated from the scavenger flotation product, and only the sand portion goes to the regrind section. The minerals in the separated slime are extremely resistant to all methods of selective separation and pose one of the major problems awaiting solution. Nevertheless, the slime shows a profit when mixed with either the lead or the zinc concentrate, except when metal prices are very low, so it is accepted as a final product for lack of a more profitable method of handling it. In contrast, the minerals in the reground sands of the

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Table I. Parral Mill Products—Second Quarter of 1951

Product	Assays							Recoveries Crude Ore Content, Pct					
	Grams		Pct										
	Au	Ag	Pb	Cu	Zn	Fe	Insol	Au	Ag	Pb	Cu	Zn	Fe
a-b—Bulk copper-lead float	7.01	1954	46.7	12.0	5.5	11.2	1.0	45.3	66.0	68.9	74.3	4.6	16.6
a—Separated copper concentrate, final	2.94	673	8.1	30.3	2.9	26.2	1.0	5.3	6.3	3.3	52.2	0.7	10.8
b—Separated lead concentrate, final	8.57	2447	61.55	4.9	6.4	5.4	1.0	40.0	59.7	65.6	22.1	3.9	5.8
c—Gravity lead concentrate, final	17.43	2296	71.4	1.5	3.9	4.2	0.7	2.9	2.0	2.7	0.2	0.1	0.2
d—Lead from zinc concentrate, final	7.35	1805	42.8	5.2	16.3	8.0	2.0	6.1	7.8	8.1	4.1	1.7	1.5
b-c-d—Total lead concentrate	8.65	2349	59.08	4.9	7.8	5.8	1.0	49.0	69.5	76.4	26.4	5.7	7.5
e—Scavenger slime product, final	2.80	435	7.0	1.1	19.0	18.6	12.0	4.1	3.3	2.3	1.6	3.6	6.3
b-c-d-e—Combined lead concentrate and scavenger slime*	7.45	1956	48.4	4.1	10.1	8.4	3.3	53.1	72.8	78.7	28.0	9.3	13.8
d-f—Zinc concentrate before deleading	1.67	274	4.5	1.2	52.6	5.5	3.4	19.7	16.9	12.1	13.6	79.9	14.9
f—Zinc concentrate after deleading, final	1.24	158	1.6	0.9	55.3	5.3	3.5	13.6	9.1	4.0	9.5	78.2	13.4
g—Tailing, final	0.26	21	0.57	0.1	0.85	2.5		28.0	11.8	14.0	10.3	11.8	62.0
								100	100	100	100	100	100

* Final lead products b-c-d-e are combined in a thickener to form a single concentrate before filtering.

scavenger product are highly amenable to separation and recovery.

The assays and recoveries for the products removed in various stages from the Parral mill circuits are listed in Table I.

One deviation from ordinary procedure is the retreatment of the final zinc concentrate for the recovery of lead. This is accomplished by depressing the zinc minerals with cyanide and zinc sulphate and then refloating to remove a lead concentrate.

Certain characteristics of the minerals in the Parral mill feed make such a deleading process necessary and advantageous. The ore supply comes from various veins that contain minerals covering a wide range of floatability. In addition, the veins that supply the major part of the tonnage have been cut by a fault which created an extensive shattered zone in which the minerals are in various stages of alteration. As a consequence, the lead minerals in the mill feed include a large proportion of easily floated clean galena, mixed with more refractory and partially altered minerals, which do not float readily. The separation of lead from zinc is complicated further by the ever-present problem of the slimed minerals, which do not respond satisfactorily to the procedure that is best suited for selective recovery of the granular portion.

It is evident then that part of the lead mineral is readily floatable and can be separated easily and cleanly from the zinc, another part has the same degree of floatability as the zinc and requires special treatment for selective separation, while the remaining and most refractory part is less floatable than the zinc and can be recovered only by a strong scavenger flotation of the zinc circuit tailing.

Such conditions preclude a reasonable recovery of the lead in a single flotation operation without floating an excessive amount of zinc. This led to experiments with a zinc concentrate deleading circuit in 1945, with such beneficial results that the procedure has been continued without interruption since that time. The advantages of the deleading process are as follows:

1—It simplifies the operation of the lead flotation circuit by making unnecessary any attempt to float the more refractory portion of the lead mineral at the expense of floating an excessive amount of zinc. Instead, a high grade lead concentrate low in zinc is floated, thereby dropping more of the zinc mineral into the lead circuit tailing for recovery in a zinc concentrate.

2—The lead mineral that floats in the zinc circuit has the same degree of floatability as the zinc mineral, and the segregation and treatment of minerals in the same restricted range of floatability favors a more effective separation. Even a slight deactivation of the zinc makes it less floatable than all of the lead mineral within this limited range. Such favorable conditions for separation cannot be established in the lead rougher circuit where the most refractory portion of the lead mineral is still present.

3—A high concentration of cyanide is used on the relatively small tonnage of zinc concentrate for maximum deactivation of the zinc, and it would not be economically permissible to use the same concentration of cyanide for the large tonnage in the lead rougher circuit to deactivate the zinc to the same degree. For example, 3.5 lb of cyanide and 2.0 lb of zinc sulphate per ton of zinc concentrate are used in the zinc deleading circuit at Parral, equal to 0.3 lb of cyanide and 0.2 lb of zinc sulphate per ton of crude ore mill feed. The high concentration of cyanide in the deleading circuit requires only one twelfth of the quantity that would be needed to establish the same concentration in the lead rougher circuit, and the cyanicides in that circuit make the cyanide less effective than when it is used on the clean zinc concentrate.

Cyanide and zinc sulphate are the only reagents used in the deleading circuit. The residual reagents in the feed to the circuit provide all the activation and froth needed to float the lead, but on occasions a small amount of alcohol frother is used to help carry the lead through the cleaner cells. Thickening and long conditioning were found to be detrimental to the separation, although some benefit is derived from a brief contact with cyanide and zinc sulphate

Table II. Deleading of Zinc Concentrate at Parral Mill

Product	Zinc Conc., Pct	ASSAYS					
		Grams		Pct			
		Au	Ag	Pb	Cu	Zn	Fe
Zinc conc.	100.00	1.67	274	4.5	1.2	52.6	5.5
Lead from zinc conc.	7.04	7.35	1805	42.8	5.2	16.3	8.0
Deleaded zinc conc.	92.96	1.24	158	1.6	0.9	55.3	5.3

Product	RECOVERIES BASED ON ZINC CONCENTRATE, PCT				
	Au	Ag	Pb	Cu	Zn
Zinc conc.	100.0	100.0	100.0	100.0	100.0
Lead from zinc conc.	31.0	46.4	67.0	30.5	2.2
Deleaded zinc conc.	69.0	53.6	33.0	69.5	97.8

to permit their complete diffusion throughout the pulp before flotation. The feed has a slight natural alkalinity of pH 7.8, and experimental additions of lime have had no noticeable effect on the separation.

As shown in Table II, the deleading procedure recovers 67.0 pct of the lead content of the zinc concentrate with a loss of only 2.2 pct of the zinc content, and based on the crude ore mill feed, see Table I, this is equivalent to a recovery of 8.1 pct of the lead with a loss of only 1.7 pct of the zinc. The percentage of gold, silver, and copper removed with the lead makes the procedure still more desirable when the zinc concentrate is sold on a schedule that pays for the zinc content only.

Table III shows the percentage of crude ore lead content in the zinc concentrate before and after the zinc deleading circuit was placed in operation. During 1942, 1943, and 1944, when lead was recovered in a single step in the lead flotation circuit only, from 6.5 pct to 7 pct of the crude ore lead content remained in the zinc concentrate, and it could be kept at that point solely by floating from 12 pct to 13 pct of the crude ore zinc content with the lead concentrate. After the introduction of zinc deleading in 1945, the lead in the zinc concentrate gradually decreased to less than 4 pct of the crude ore content, with at the present time the inclusion of less than 10 pct of the crude ore zinc content in the combined lead products, see Table I. This represents a 2.5 pct increase in lead recovery in conjunction with a 2.5 pct decrease in zinc recovery in the lead concentrate, and at the same time, the zinc dropped from the lead concentrate improved the recovery in the zinc concentrate by an equal amount.

Table III. Crude Ore Lead Content in Zinc Concentrate After Zinc Deleading Circuit

Procedure	Year	Crude Ore Lead Content Remaining in Zinc Conc., Pct
Zinc conc. not deleaded	1942	6.5
	1943	7.0
	1944	6.7
Deleading started July 30th	1945	4.8
Zinc conc. deleading	1946	4.3
circuit in continuous	1947	4.1
operation	1948	4.3
	1949	3.9
	1950	3.6

The concentrating plant of the Asarco's Santa Barbara unit is located near the village of Santa Barbara, which lies in the foothills of the Santa Barbara mountain range in the southcentral part of the State of Chihuahua at an elevation of 6300 ft. The prevailing rock in the district is shale which has been intruded by rhyolite and basalt dikes. This shale is cut by numerous veins that have been mined since early colonial times. The work of the Spaniards was limited to the oxide portion of the veins, which extends several hundred feet below the outcrops except in those areas where erosion has been excessive. At the present time, mining is confined to the deeper sulphide areas of the veins. Ores from five different mines make up the mill feed, and the principal minerals are lead, mainly occurring as galena, copper in the form of chalcopyrite and to some extent as chalcocite, and zinc as a marmatitic sphalerite, all associated with pyrite and some gold and silver in a siliceous gangue.

For a number of years, and till the middle of 1949, the flowsheet of the mill followed the conventional lines of most plants practicing selective flotation of lead and zinc. As additional features, this flowsheet included—and still includes—gravity concentration of the lead liberated in the primary grinding circuit, a lead-copper separation process by means of sulphurous acid, and a scavenger circuit following the zinc float.

In view of the fact that most of the Santa Barbara ores presently treated show varying degrees of alteration and oxidation of the mineral surfaces and, with few exceptions, cannot be classed as clean ores as regards selectivity and floatability of the lead and zinc constituents, they require strong flotation conditions in the lead circuit for optimum lead-copper recoveries and the subsequent production of a zinc concentrate of acceptable grade. It is obvious that this procedure tends to float a good deal of zinc into the lead concentrate, especially when the ore is very refractory. In the past, these zinc losses, and the resulting comparatively low grade lead concentrates, were tolerated as long as the value of the refractory lead recovered was greater than the value of the zinc that had to be taken into the lead concentrate and thus was lost as a pay product.

Early in 1949, however, the amounts of refractory material in the various ores delivered to the mill increased to such an extent that it became more and more difficult to produce marketable lead and zinc concentrates and maintain recoveries. In spite of the customary strong flotation conditions in the lead circuit, a good deal of the lead and also copper dropped into the zinc section and floated there under the activating influence of the copper sulphate but to the detriment of the grade of the zinc concentrate produced. Attempts to retard this lead in the zinc circuit with potassium bichromate and to float it in the scavenger circuit were not successful. Therefore it was decided to try a deleading retreatment of the final zinc concentrate along the lines developed in the Parral mill. When it was possible to show that the lead could be floated away successfully from the zinc after the latter had been depressed with cyanide and zinc sulphate, this deleading operation became part of the regular flowsheet in July 1949. The immediate result of this procedure was a noticeable improvement in the zinc concentrate grade through the removal of considerable amounts of lead, copper, and iron minerals from the original zinc concentrate with relatively low losses of zinc in the refloat lead concentrate. In turn, the lead and copper values thus recovered and added to the original lead-copper concentrate correspondingly increased the recoveries in the lead and copper circuits.

As soon as the deleading operation had been firmly established, it became possible to change the conditions in the original lead float. Instead of attempting to recover as much lead as possible in the lead circuit, the operators were instructed to disregard the lead section tailings, to produce a high grade lead concentrate of minimum zinc content and to allow the refractory lead minerals to pass into the zinc circuit for their subsequent recovery in the refloat circuit.

The overall effect of the refloat operation and typical results obtained are shown in Table IV.

Table IV brings out the fact that the refloat operation not only produces a relatively high grade final zinc concentrate and a lead-copper concentrate of acceptable grade, but also recovers 8.4 pct of the total lead in the mill feed with a loss of only 2.1 pct of the total zinc. The value and effectiveness of the refloat circuit are thus clearly emphasized. At the same time assays of the zinc concentrate before the deleading reflect the operation of the original lead circuit and indicate the amounts of lead and copper that were allowed to drop into the zinc section. It is obvious that, without a deleading circuit, a zinc concentrate assaying 5.4 pct lead, 1.6 pct copper, and only 50.2 pct zinc would not be produced for financial reasons, even if such a concentrate were acceptable to the smelter, and that the lead circuit would be operated so as to produce a greater recovery of the lead and copper values, which naturally means a lower grade concentrate and a higher loss of zinc.

It is to be understood that the figures in Table IV represent average results only. It will be readily seen that the overall value of the entire refloat operation increases as the refractoriness of the ore treated increases, whereas with cleaner ores the results are less spectacular.

In the zinc deleading operation as practiced at Santa Barbara the primary zinc is double-cleaned in a ten-cell No. 24 Denver Sub-A machine, and then is pumped to an eight-cell No. 24 Denver Sub-A machine. A rougher lead concentrate is taken from cells No. 3, 4, 5, and 6 to be cleaned in No. 2 cell and recleaned in No. 1 cell. The rougher concentrate from the last two cells of the machine is returned to No. 4 cell. The double-cleaned lead concentrate joins the original lead-copper concentrate for further treatment in the lead-copper separation section, the tails of the machine constituting the final zinc concentrate.

Principal reagents added ahead of the pump are cyanide, zinc sulphate, and lime. Minor amounts of xanthate, cresylic acid, and of an alcohol frother are added at the machine as needed. Approximate reagent requirements per ton of primary zinc concentrate feed to the refloat section are: 2.3 lb cyanide, 4.4 lb zinc sulphate, 3.2 lb lime, 0.03 lb xanthate, and 0.01 lb each cresylic acid and Du Pont frother B-22.

Cost of operating the section, including reagents, power, and maintenance, amounts to approximately 8¢ to 10¢ per ton of crude ore milled.

Both cresylic acid in the rougher and alcohol frothers in the cleaner section have been found necessary and, if judiciously used, helpful. Obviously, overfrothing is to be avoided not only in the refloat circuit but also in the primary zinc rougher and cleaner section. The importance of the latter factor was demonstrated in the plant some time ago when Aerofloat No. 242 was tried in the zinc rougher float as a substitute for the standard xanthate collector. It was found that the frothing properties of this reagent, even if used in less than the necessary amounts, were so pronounced as to make the refloat circuit uncontrollable in a very short time.

Table IV. Overall Effect of Refloat Operation and Typical Results

	ASSAYS					
	Grams		Pct			
	Au	Ag	Pb	Cu	Zn	Fe
Zinc Conc.						
Before deleading	1.2	294	5.4	1.6	50.2	7.5
After deleading	0.8	171	1.1	1.1	54.5	7.2
Refloat lead conc.	4.6	1364	42.6	6.0	12.9	10.2
	RECOVERIES IN REFLOAT LEAD CONC., PCT					
	Au	Ag	Pb	Cu	Zn	Fe
Based on crude ore mill feed	2.9	6.8	8.4	8.7	2.1	2.4
Based on deleading section feed	39.6	47.9	81.4	38.7	2.6	14.0

Table V. Comparison of Parral and Santa Barbara Deleading Results

	ASSAYS					
	Grams		Pct			
	Au	Ag	Pb	Cu	Zn	Fe
Zinc conc. before deleading						
Parral	1.7	274	4.5	1.2	52.6	5.5
Santa Barbara	1.2	294	5.4	1.6	50.2	7.5
Zinc conc. after deleading						
Parral	1.2	198	1.6	0.9	55.3	5.3
Santa Barbara	0.8	171	1.1	1.1	54.5	7.2
Lead conc. from zinc conc.						
Parral	7.4	1805	42.8	5.2	16.3	8.0
Santa Barbara	4.6	1364	42.6	6.0	12.9	10.2
	RECOVERIES IN REFLOAT LEAD CONC., PCT					
	Au	Ag	Pb	Cu	Zn	Fe
Based on deleading section feed						
Parral	31.0	46.4	67.0	30.5	2.2	10.3
Santa Barbara	39.6	47.9	81.4	38.7	2.6	14.0
Based on mill feed						
Parral	6.1	7.8	8.1	4.1	1.7	1.5
Santa Barbara	2.9	6.8	8.4	8.7	2.1	2.4

pH Control

For reasons as yet unknown, the great amount of laboratory work done to study the effect of lime has, so far, failed to furnish the necessary data on which definite conclusions may be based, and a good deal more work will be required to answer the fundamental question as to whether lime should be used or not. Results obtained in the plant show that, without lime, the operation of the refloat section becomes rather unstable and erratic, a fact also borne out in some of the laboratory series where the absence of lime led to poor lead-copper recoveries because of a pronounced lack of selectivity. On the other hand, quite a few test series showed a normal float and an acceptable lead-zinc separation when no lime was used. Notwithstanding this erratic and as yet unexplained behavior of the different mill pulps tested, certain general trends resulting from the use of lime may be summarized as follows. Up to a pH value of 9 or 9.5, recoveries of lead and copper are better than those obtained at a pH value of 8 which is the natural pH of the primary circuit in the plant. As the pH value of the refloat is raised above 9.5, the floatability of the lead and copper begins to decrease, and in the case of the copper, sometimes so rapidly as to lead to a definite rejection. Zinc rejection improves as the pH value of the float increases. From the few data presented it is obvious that the use of lime in the refloat circuit is, if anything, critical, and requires a careful study and balancing of all the factors involved in each individual case.

A comparison of the Parral and Santa Barbara deleading results is shown in Table V.

The results obtained at Parral and Santa Barbara indicate that a zinc concentrate deleading process may be a valuable means of improving concentrate grades and recoveries wherever a poor lead-zinc selectivity poses a major metallurgical problem.

Screened Ore Used for Fine Grinding at Lake Shore Mines

by Bunting S. Crocker

PEBBLE grinding at Lake Shore is not a temporary wartime substitute. The tube milling plant, with a 1000 ton per day capacity, grinds a hard siliceous ore to 90 pct − 325 mesh. The plant, prior to using pebbles, was consuming 4.3 lb of 1¼-in. grinding balls per ton of ore, which amounted to 785 tons of balls per year. At September 1951 prices, $132.60 per ton, this steel cost amounted to $104,400 per year, or $0.285 per ton milled. By the Lake Shore method of substituting screened rock for this steel, all of this cost is saved. This is one of the major economies in Lake Shore mill practice. Regardless of the ultimate price of grinding balls, a change back to steel balls is not considered. The present pebble plant is more flexible than a steel ball plant and equally efficient. For example, if it is desired to change the size of grinding media, the pebble charge in any mill can be changed completely in 4 to 5 days, as against 76 days to change completely a 1¼-in. steel ball charge.

To change pebble size it is necessary only to change two sets of screens and clean out the rock feed storage bin. This will take 8 to 10 days as against 3 to 4 months to clean out the customary supply of grinding balls kept on hand. Also it has not always been possible to purchase all desired sizes of balls at any price.

In an analysis of savings effected by the use of the pebble mills, the flexibility of the Lake Shore grinding plant should be discussed, as it has a direct bearing on these savings. The plant has always used several units to handle tonnage rather than sending all the tonnage through a single unit. This principle may result in using small diameter mills, but no objection to that is seen. At no time has any advantage been found in cost per ton ground in large diameter mills. Both the capacity and the power of any mill varies as the diameter raised to the 2.6th power. Consequently large or small mills are equally efficient, and a plant should be designed to use as many units or combination of units as is consistent with reasonable operating practice. Mills under 5 ft diam are harder to reline, etc.

To define the case at Lake Shore, 2600 tons per day formerly were milled in seven 7x6-ft ball mills and twelve 5x16-ft (and two 6x16-ft) tube mills.* This gave an excellent test plant and an extremely efficient one. In this plant the ratio of ball mills to tube mills was 1:2. When the much cheaper pebble mills were substituted for the tube mills, this ratio was changed to one ball mill to four pebble mills to take the greatest possible advantage of the cheaper operating mill, i.e., the pebble mill.

This flexibility without loss in efficiency has been an important item in the cost savings.

It is interesting to note that the use of pebbles for fine grinding was proved first in the laboratory in a 12-in. ball mill. In fact, since 1934 all testing on − 8 mesh material has been done in this 12-in. mill.

* Throughout this paper, the term tube mill is used in referring to a mill filled with small steel balls and, incidentally, a mill in which the length is at least two times the diameter.

Scope of the Tests

A paper on fine grinding at Lake Shore Mines was published in July 1940.[1] This paper covered 7 years of intensive research on fine grinding as well as sizing methods and equipment, plant scale grinding tests on 5x16-ft tube mills with and without grate discharges both with 1¼-in. and ¾-in. balls, the use of laboratory mills to evaluate plant changes, and several reports on classifiers and classification. In the following July the addendum report[2] was added in which the idea of series-circuit grinding was introduced, and the results of running five stages of tube mills and bowl classifiers were shown.

Since 1940, the ball milling end of the plant has been altered extensively as a result of tests on the use of 3½-in. rods in 7x6-ft mills and the use of the Tyler repulping screen with from 7 to 14 mesh screens. These tests are lengthy and may be covered in a separate report later.

The scope of this report is confined to ore ground by rod milling and ball milling until it passed through an 8 mesh Tyler Ty-rod screen. The −8 mesh screen undersize then was pumped to a primary bowl classifier in open circuit and the sands from the bowl sent to the primary pebble mills, see Fig. 1. In the pebble mill circuit the ore is ground to 90 pct −325 mesh (24 pct + 28 microns). In studying the flowsheet, attention should be paid to the efficiency of the classification equipment used. The Tyler repulping screen is an efficient machine on the 8 to 10 mesh separation, and the bowl classifier is equally efficient at the 325 mesh separation. Efficient classification is a necessity for series-circuit stage grinding.

The ore is hard siliceous porphyry, 60 pct SiO_2, 80 pct insoluble. Its grindability at different meshes has been shown near the top of the list in F. C. Bond's grindability tests.[3] Lake Shore is not shown, but an adjacent mine with identical ore, Wright-Hargreaves, is.

Reasons for the Changeover

Since 1936 grinding balls have been rising steadily in price with no sign of stopping. For a mill that used 4.5 to 5.2 lb of grinding balls for every ton of ore ground this rise represented an alarming increase in grinding costs. In many cases the quality of the grinding balls fell off as the scrap steel became more difficult to obtain. The ratio of tube mills to ball mills increased with the use of the Tyler repulping screen in the ball mill circuit. Originally only 3.0 lb of balls per ton were used in the tube mills, and this

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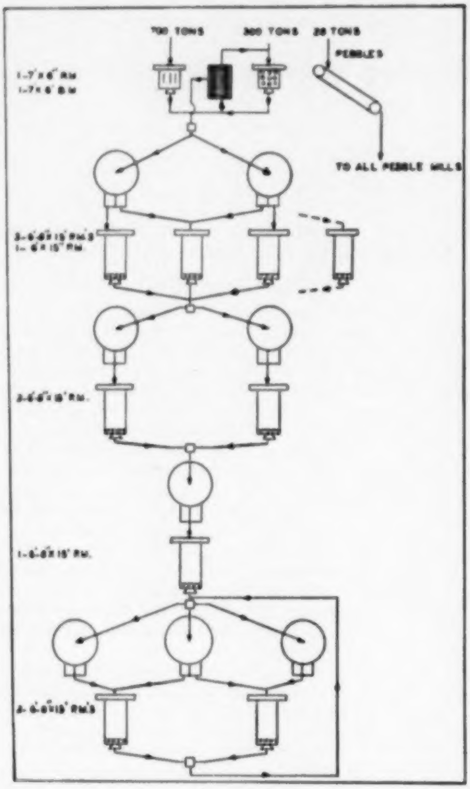

Fig. 1—Grinding flowsheet.

figure had risen 43 pct to 4.3 lb per ton by 1949 because of above-mentioned factors, see Fig. 2.

Inability to obtain ¾-in. grinding balls in the summer of 1948 was one of the final factors in the decision to convert. Lake Shore had pioneered the use of this small ball and had made a saving of 23½ pct more capacity at 7 pct more cost. Not only did the manufacturers refuse to supply the small ball, but they indicated that when the scrap metal situation eased up they would put the ball back on the

Fig. 2—Price of 1¼ x ¾ in. grinding balls in Canada by years.

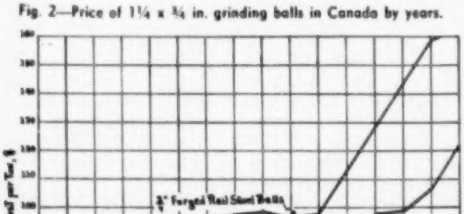

Table I. 10-G Slugs Vs 10-G Balls

Mesh	Feed No. 9 Secondary Class Sands	15-Min Grind with Slugs	15-Min Grind with Balls	10-Min Grind with Slugs	10-Min Grind with Balls
8	0.1	0.2	0.3	0.2	0.2
10	0.6	0.5	0.7	0.5	0.6
20	2.0	1.8	1.5	1.4	1.4
35	4.5	1.6	1.0	1.5	2.4
65	18.3	9.0	9.9	10.0	9.4
150	26.3	21.1	20.7	20.4	16.1
+325	38.6	38.5	39.2	35.0	39.9
−325	9.6	27.3	26.7	31.0	30.0
	100.0	100.0	100.0	100.0	100.0

market at a price of $150 to $160 per ton. At this price differential, the economic use of small balls is prohibited. United States manufacturers of grinding balls have always maintained a much greater price differential for various sizes of grinding balls than Canadian manufacturers, and as a result the use of small balls has been discouraged.

It should be mentioned that Lake Shore laboratory tests had always shown, for the final stage of the grinding plant, 5 to 10 pct −65 mesh feed to final mills, that ½-in. grinding balls were 22 pct more efficient than ¾-in. balls. However, Lake Shore has been unable to interest anyone in manufacturing them.

It has also been found that ¼-in. grinding balls were not so satisfactory as ½-in., and ⅛-in. balls failed entirely on all but −250 mesh material.[4] Laboratory and plant tests proved conclusively that the same size of ball could be used with the same efficiency in mills between 1-ft and 6 ft 8-in. diam.

The loss of the small ball also spoiled the application of the theory behind the multistage grinding plant. If the size of grinding media could not be controlled in successive stages, part of the efficiency of stage grinding would be lost.

The above-mentioned points were important in themselves and more important as a whole in their effect on the company's attitude toward research that sought a permanent solution.

It is necessary that the reason for the use of ore as a grinding media be explained. The basic idea of pebble grinding is very old. Danish pebbles and pebbles from local quarries have been used from time to time, but usually to keep the circuit free from iron or for special grinding, seldom to cut costs, as they were generally more expensive to use in this country. They have, however, been used extensively in West Australia.

Many prominent metallurgists from South Africa who visited the plant told the company that the pebbles from the Rand Conglomerate were an excellent material for grinding. However, the necessity for cheap native labor both to hand pick the pebbles and to cull the loads from time to time and remove unsuitable fine material would make such a process impossible in Canada, even if the ore itself should prove to contain rock suitable as pebbles. The timely visit of Dr. Jackson in 1948 helped to dispel some of this feeling of pessimism. He reported that many mills had stopped the practice of culling the loads and generally encouraged the idea of trying ore as a grinding media.

Test on Ball Shape in Fine Grinding

Before Dr. Jackson's visit, when it was realized that the ¾-in. ball would no longer be available, a cheap steel substitute was sought. One early idea was

Table II. Tungsten Carbide Balls Vs Steel Balls on Fine Feeds

Mesh	Feed	Tungsten Carbide 1¼ In. Load 400 Rev.[a]	Steel 1¼ In. Load 667 Rev.
3			
8	0.1		
10	5.7	0.2	0.2
20	21.6	0.2	0.2
35	25.6	0.5	0.5
65	20.9	11.0	10.7
150	13.5	31.2	30.2
270	6.9	19.9	19.9
−270	5.7	37.0	38.3
	100.0	100.0	100.0
Cum + 10	5.8	0.2	0.2
Cum + 35	53.0	0.9	0.9
Cum +150	87.4	43.1	41.8

[a] Mill speed 60 rpm.

to use ¾-in. and ½-in. rods cut into cylinders, and another idea was to use the punchings from holes on the ends of rails. It was reasoned that grinding media of this type might be inefficient but because of its low cost might prove economical. However, when these odd shapes were tested for fine grinding it was found that they ground as efficiently as a ball of the same weight. Old Erickson bits had been tested previously as a substitute for 1¼-in. balls, and although the test was not as long as desired, the mills kept grinding through the run with no marked loss of efficiency.

A test was made in the 12-in. Haultain mill using a charge consisting of 10-g punchings from the Algoma Steel Co. These punchings were approximately ⅝-in. diam, by ¼-in. thick. Balls of equal weight, slightly over ½-in., were taken from a 1¼-in. ball load and were all reasonably spherical. Since both grinding media were made of steel, equal weight and equal mean size would be the same. The results of the tests on odd shapes are shown in Table I.

On these fine sands, the odd-shaped flat slug ground as well as a spherical ball. Both grinding media had the same weight and therefore the same mean size and performed the same work. Both media failed to grind the 10 and 20 mesh material, and larger slugs would be necessary for plant grinding. Feed up to 10 mesh was chosen deliberately to determine the effect of shape on the top meshes.

The fact that in grinding −8 mesh sand the grinding media need not be too spherical helped greatly to eliminate some of the worries about Lake Shore ore. Characteristically this ore breaks with one long dimension, particularly the product from the Symons cone crusher. Broken ore at Lake Shore did not appear to be suitable grinding media.

However, it should be remembered that every conceivable shape of grinding ball, i.e., concave balls and even cubes, has been patented. It is thought that much of the confusion in the industry regarding the shape of the grinding ball probably came from the failure of operators to use the right correction factor for liner wear when testing various media in small diameter mills.

Tests on Tungsten Carbide Balls

A test was run in the Lake Shore Mines experimental laboratory on the use of various sizes of tungsten carbide grinding balls. These tests were made possible through the courtesy of D. Weston of Aerofalls Mills Ltd., who loaned the ball charges. The tungsten carbide ball tests showed clearly that balls will grind in capacity according to their specific

Table III. Tungsten Carbide Balls Vs Steel Balls on Coarse Feeds

Mesh	Feed	Tungsten Carbide 1¼ In. Load 1900 Rev.	Steel 1¼ In. Load 1500 Rev.
3	17.8	9.2	9.4
6	36.3	14.9	14.4
10	16.3	1.1	1.2
20	10.8	0.4	0.4
35	5.7	0.4	0.4
65	3.4	0.7	0.6
150	2.2	3.4	1.8
−150	7.5	69.9	71.8
	100.0	100.0	100.0
Cum + 10	70.4	25.2	25.0
Cum + 35	86.9	26.0	25.8
Cum +150	92.5	30.1	28.2

Table IV. Effect of Surface Hardness in Grinding

Mesh	Grind Head	1¼ In. Steel, 667 Rev.	1¼ In. Tungsten Carbide, 400 Rev.	1¼ In. Stellite, 600 Rev.
+ 8	1.1	trace	trace	trace
+ 10	11.2	0.3	0.2	0.3
+ 20	42.1	0.4	0.4	0.5
+ 35	23.2	1.2	1.8	1.2
+ 65	10.9	23.5	25.2	22.9
+150	6.0	23.4	23.4	23.3
+325	4.6	21.1	20.7	21.3
−325	0.9	30.1	28.3	30.5
	100.0	100.0	100.0	100.0

Table V. Size of Ball or Pebble Required for Fine Grinding

Mesh	Classifier No. 4and 5 Primary Sands	Classifier No. 7 and 8 Secondary Sands	Classifier No. 12 Tertiary Sands
+ 6	0.4		
+ 10	7.0	0.5	0.2
+ 20	26.4	2.3	0.5
+ 35	23.4	5.8	0.7
+ 65	16.3	16.7	5.5
+150	11.6	31.7	31.7
+270	6.2	24.3	39.1
−270	6.5	18.7	22.3
	100.0	100.0	100.0
Cum + 10	7.4	0.5	0.2
Cum + 35	50.2	8.6	1.4
Cum +150	67.3	57.0	38.6

Table VI. First Laboratory Pebble Test

	Head Primary Sands	1¼ In. Steel Load, 300 Rev	1¼ In. Pebbles, 1300 Rev	Pebble Analysis[a] Size	No. Pieces	Pct Wt
Capacity rating, pct		100	33	+1½	12	15.8
+ 6 mesh	0.4			+1	76	57.2
+ 10 mesh	1.3	0.3	0.2	+¾	85	23.8
+ 20 mesh	1.8	0.2	0.2	+½	8	0.9
+ 35 mesh	3.6	0.3	0.3	−½		2.3
						100.0
+ 65 mesh	17.5	5.3	4.9			
+150 mesh	36.8	31.7	29.0			
+325 mesh	32.6	37.4	36.6			
−325 mesh	6.0	24.9	28.8			
	100.0	100.0	100.0			
Pct new −325 produced		18.9	22.8			
Capacity rating, pct		100	120			

[a] Average weight per piece is 48 g. equivalent to a pebble sphere 1.28 in. in diam or equivalent to a steel sphere 0.9 in. in diam.

Table VII. Diam of Balls of Different Material Having the Same Weight

Weight, G	Diam of Steel Ball, In. (7.8 Sp Gr)	Diam of Ore (2.66 Sp Gr)	Diam of Tungsten Carbide (13.1 Sp Gr)
3.52	0.375	0.53	0.32
8.37	0.50	0.72	0.42
16.3	0.625	0.89	0.53
28.2	0.75	1.07	0.63
43.4	0.875	1.24	0.73
66.9	1.00	1.42	0.84
131.	1.25	1.77	1.05
226.	1.50	2.14	1.26
347.	1.75	2.48	1.46
535.	2.00	2.86	1.68
1807.	3.00	4.27	2.53
4282.	4.00	5.72	3.36
8364.	5.00	7.17	4.20

gravity, see Tables II and III. A tungsten carbide ball of the correct size will handle 68 pct more tonnage than a steel ball of the same weight. The correct size of ball must be determined first with regard to the size of material being ground. This means that the Lake Shore pebble would be expected to handle only 34 pct of the tonnage of steel balls. This realization was an important factor in the ultimate success of pebble testing at Lake Shore. Early scouting tests on grinding with a rock grinding media had never shown any promise and had been abandoned. They had been tested at too high a feed tonnage and naturally showed little grinding under these conditions. The main value of the tungsten carbide test was that the balls themselves proved too expensive for general use, although the ⅜-in. tungsten carbide balls showed great efficiency and might be used in special applications.

The fact that the pebble mills would have less capacity than steel mills was no handicap in itself, as the mill formerly handled 2600 tons and was now milling 1000 to 1200 tons, and there were several idle 5x16-ft mills and one 6x16-ft mill that could be pressed quickly into service.

Tests were made in the 12-in. Haultain grinding mill with loads of 1¼, 1, and ¾-in. balls according to the Prentice formula. The same volume of balls was used in each case. Thus the tungsten carbide load was 1.68 times as heavy as the steel load with 13.1:7.8 to 1.68:1. sp gr.

Several grinds were run with different lengths of time, i.e., varying revolutions of the mill with the speed of the mill constant, 80 pct critical. It was established finally that the two loads would do identical grinding when the steel load was run 68 pct longer than the tungsten carbide load. Thus the tungsten carbide load has 68 pct more capacity than the steel load, see Table II. Another test on coarse feed was run to see if the heavier ball would handle coarser material. When the tungsten carbide mill is handling 68 pct more feed than the steel mill, both mills have equal ability to handle the top meshes, see Table III.

In this test, balls of equal size were used rather than of equal weight, and at a capacity differential of 68 pct the lighter steel ball is slightly more efficient. A steel ball of equal weight, 1½-in., probably would have shown the same capacity as the tungsten carbide under the above conditions. This is a delicate problem and one which must be investigated carefully when testing balls of different specific gravity.

Some interesting test work also was done with ⅜-in. tungsten carbide balls as compared to ⅜-in. and ¾-in. steel balls. With steel balls running at proper gravities for good grinding, the ½-in. grinding ball is the lower limit. The ⅜-in. ball is slightly off and the ¼-in. ball a definite failure. The tungsten carbide ball, which is 68 pct heavier, retains the advantage of its extra surface and showed 80 pct more capacity than ⅜-in. steel and 96 pct more capacity than ¾-in. steel.

Surface Hardness in Grinding

Tungsten carbide grinding balls have a surface hardness much greater than rail steel balls. To check if this superior hardness had anything to do with the increased grinding capacity, a test was made using steel balls, tungsten carbide balls, and stellite balls. The stellite is 11 pct heavier than steel but has a much harder surface, comparable to that of the tungsten carbide. The time of grind, and thus the capacity, in each test was varied inversely as the specific gravity of the grinding media so that the final grinds would all be the same, if capacity varies according to specific gravity and surface hardness has no effect.

The results of this test, given in Table IV, show that surface hardness has no effect. The tungsten carbide ball is slightly off its capacity rating since the same size ball was used instead of the same weight. A 1.05 in. tungsten carbide ball is the equivalent in weight of a 1¼-in. steel ball. This fact was not appreciated at the time the test was made. It became more apparent with greater difference in specific gravity between the grinding media. The 11 pct difference between steel and stellite would not have any appreciable effect.

The sand analyses given in Table V shows the type of feed being handled by each stage of milling.

Tests at Lake Shore show that the most efficient

Table VIII. The Effect of Pebble Size on the Grind, July 1948

	Mesh	1 Final Class Sands	2 ¾ In. Pebbles with 11 Pct −¼ In. in Load, 1250 Revs	3 Same Load with −¼ In. Removed, 1250 Revs	4 50 Pct −1 In. + ¾ In. 50 Pct −¾ In. + ½ In., 1250 Revs	5 All −1 In. + ¾ In., 1250 Revs	6 −¾ In. Steel Load 500 Revs
	+ 10	0.1	—	—	—	—	—
	+ 20	0.7	0.5	0.5	0.1	trace	0.1
	+ 35	1.6	2.2	1.5	1.0	0.5	0.1
	+ 65	8.6	11.0	8.8	7.8	6.6	3.0
	+150	19.0	15.4	13.5	15.2	15.8	13.7
	+325	54.0	42.7	42.0	43.6	44.5	47.5
	−325	16.0	28.2	33.7	32.3	32.6	35.6
		100.0	100.0	100.0	100.0	100.0	100.0
Cum.	+ 35	2.4	2.7	2.0	1.1	0.5	0.2
Cum.	+150	30.0	29.1	24.3	24.1	22.9	16.9
Cum.	+325	84.0	71.8	66.3	67.7	67.4	64.4

ball for primary sands is 1-in. to 1¼-in.; for secondary sands, ¾ to ⅞-in.; for tertiary sands, ½-in. balls.

Early Laboratory Tests

Most of the laboratory tests with the 12-in. mill are *tonnage tests.* At least four varying times of grind are made, each at the correct pulp density, with each different variable being tested. The variable may be 1—coarse or fine sand feeds, 2—different size of grinding media, or 3—different types of grinding media. The time of grind has been tested against the operating plant, and time of grind can be related to tonnage per day through a 6 ft 8-in. x 15-ft pebble mill or a 5-ft x 16-in. tube mill. The log of this tonnage then is plotted against the log of the percentage reduced† of any reliable mesh in the screen analysis. The resulting plot has been found to be a straight line with the laboratory mill and with plant pebble mills, tube mills, ball mills, and rod mills. This relationship holds with normal variations in feed analysis so that tests done at different times can be compared. This relationship has proved of tremendous help in checking laboratory work and in designing alternate layouts or new plants. It has been used successfully for the past 10 years. To describe the method fully is a report[3] in itself, and it is simply mentioned here that such a method is used. Some illustrative tests are quoted in this report, but plant design is based on the more detailed methods mentioned above.

† Percentage reduced =

$$\frac{\text{cum pct + any reliable mesh in the feed} - \text{cum pct + same mesh in the discharge}}{\text{cum pct + same mesh in the feed}}$$

Capacity and Size of Pebbles

After Dr. Jackson's visit and with the capacity ratings from the tungsten carbide tests available, encouraging grinds on the first half dozen laboratory tests using ore as a grinding media were obtained, see Tables VI and VII.

This first pebble load produced a finer grind than the 1¼-in. steel load, which was encouraging at the time. Reviewing the test now, it is found that 1—The pebble load chosen was of approximately equal size rather than of equal weight. A 0.9-in. steel load is known from the 1940 report to be 14 pct stronger on this type of sand feed than 1¼-in. balls. The size distribution of this load indicates that it is slightly superior to a 0.9 steel load. 2—The test was run at 1500 rev instead of 1470 rev for 34 pct capacity. Thus when all corrections are made, the pebble load at a rated capacity of 34 pct is at least as good as the 1¼-in. steel balls.

These tests indicated that a large pebble of the same weight as a 1¼-in. ball would grind satisfactorily. The next question was whether the work of the ¾-in. grinding ball with fine pebbles could be duplicated. The tests in Table VIII were run to check this point.

These tests showed that if the pebble size is kept above ¾ in. to ⅝ in. a good grind can be obtained (compare cols. 2 and 3). The −¾-in. steel load was run at 500 rev instead of 417 rev for its proper capacity rating. When corrected graphically for its extra time of grind, it checks the pebble of the same weight almost exactly, i.e., all −1 in. +¾ in.

Within a month of the start of the laboratory testing, the conclusion was reached that ore could be used as a grinding media if a cheap method could be found.

When the idea of grinding with rough screened rock was discussed first, it was felt that the broken rock would tend to chip, producing tremendous quantities of 3, 4, and 6 mesh material which would be too large for the pebbles to handle and would thus build up in the mill load and also might tend to block the grate discharge. Several tests were run which demonstrated that this would not happen. Rock loads ground for 24 hr showed smooth pebbles and fine slimes with a noticeable absence of intermediate sizes.

It was noted that the heaviest pebble consumption occurs during the first 2 hr, but the original pieces are not chipped into large intermediate size pieces. After 2 hr the pebbles were observed to be well rounded. During the next 24 hr the pebble consumption per 2 hr period was almost constant and only about 20 pct of the consumption during the first 2-hr period. These tests have been confirmed by over 2 years' operations in 6 ft 8-in. diam mills. There is no problem of this nature on Lake Shore ore on 5 ft, 6 ft, and 6 ft 8-in. grate discharge pebble mills.

Early Tube Mill Tests

High Discharge Pebble Mills: The first plant scale tests were made in No. 14 tube mill on Oct. 8, 1948. The 18 tons of mixed 1¼-in. and ¾-in. steel grinding balls were dumped out and 6.0 tons of approximately −2½ in. +¾-in. screened rock were put in. (The lighter rock resulted in the mill speed increasing slightly from 30 rpm to 31 rpm.) This mill was run as a high discharge pebble mill for several months with a wide variety of results. The following conclusions were reached.

1—Pebble mills require some sort of lifter bars to obtain correct pebble action. The mill was run at high speed 34 rpm (96.6 pct critical) but did not approach the effectiveness of lifter bars. Four lifter bars seemed as effective as seven, but it was considered good practice to have some lift on at least every second liner.

2—For best grinding, the mill had to be run with

Table IX. Summary of Preliminary Testing with Pebbles in 5x16 Ft High and Low Discharge Mills

Period	1948	Avg Hp	Hp Minus No Load[a]	Capacity[b] Rating	Hp Per Capacity	Speed	No. of Lifter Bars	Discharge Level
5'x16'	Steel ball mill	182	162	100	1.62	30	0	high
1st	Oct. 7-15	59.0	39.0	24.5	1.59	31	0	high
2nd	Oct. 16-28	57.6	37.6	21.9	1.71	31	0	high
3rd	Oct. 29-Nov. 4	60.6	40.6	25.3	1.61	34	0	high
4th	Nov. 5-14	73.5	53.5	31.4	1.70	34	4	high
5th	Nov. 15-24	80.0	60.0	33.3	1.80	34	7	high
6th	Nov. 26-28	78.0	58.0	33.0	1.75	31	7	high
7th	Nov. 30-Dec. 15	98.5	78.5	47.8	1.64	31	7	low
9th	Feb. 8-15	97.6	77.6	47.5	1.63	31	0	low

[a] No load — shell only 20.
[b] As determined by tonnage tests in the plant.

Fig. 3—Pinion drive of original 5x16 Allis-Chalmers tube mill.

more dilution than the steel mills, with 1.550 to 1.650 sp. gr. (56 to 65 pct solids) found best on fine feeds as opposed to 1.650 to 1.750 (65 to 70 pct solids) for steel mills.

3—High discharge pebble mills were a nuisance to operate. Changes in gravity would put the fine pebbles through the discharge trunnion and into pumps, launders, etc.

4—The capacity of the mill was spotty, varying from 22 pct to a maximum of 33 pct of the capacity of a steel mill. It took careful operating to maintain 33 pct capacity at all times.

5—The horsepower of the mill went up and down with the capacity. At all times the amount of finished material produced per horsepower was the same as with the high discharge steel mill, i.e., the total horsepower minus the power to drive the empty shell, etc. The *no-load* power is such a high percentage of the total in a test of this kind that it is necessary to make this correction to arrive at a fair power comparison between steel and pebble mills, see Table IX. Later on the 6 ft 8-in. mills the motors are properly loaded, and no correction is necessary.

All in all, high discharge pebble mills did not seem an attractive proposition, and Lake Shore Mines hesitated to try to run such a plant.

Low Discharge Pebble Mills: The pebble load was transferred from No. 14 mill to No. 13 mill on Nov. 30, 1948. No. 13 mill had been a regular low discharge primary tube mill using 1¼-in. balls and had a grid and scoop discharge.* The grid had 3/16-in. openings. This mill then was run as a low discharge pebble mill for 2 years and is still in operation.

Low discharge pebble mills draw 40 pct more power and have 40 pct more capacity than high discharge (or trunnion discharge) pebble mills. They are also much easier to operate, maintaining a steady power reading and capacity provided, of course, that the pebble load is kept at or near the center line of the mill.

This tremendous difference between high and low discharge pebble mills is in sharp contrast to the capacity change for steel mills, which is 27 pct at Lake Shore on these same mills. Apparently the 2.7 sp. gr. pebble has difficulty in grinding in high discharge mills where the pulp gravity is 1.650. This same capacity differential was found at the Rand in South Africa 20 years ago, according to C. W. Dowsett, who gave the writer the figure before the tests were run at Lake Shore. This would explain why all the South African fine grinding pebble mills are provided with grates and low discharges.

Grates with ⅜-in. openings were tried during the tests and found to be slightly too large for the size of the pebble in use at that time. The original steel grate opening of 3/16 in. was satisfactory. Later ¼-in. slots were used.

Pebble Consumption

On a rock feed with an average weight of 120 g per piece and with a screen analysis of 17 pct + 2 in., 68 pct + 1½ in., 15 pct + 1 in., the following pebble consumption figures were obtained: Without lifter bars on a 5x16 ft high discharge pebble mill, the daily consumption was 927 lb per day. Without lifters the load could be observed through the discharge trunnion to be riding only part way up one side. This resulted in the pebbles wearing into exaggerated flat shapes. When lifter bars were put in, the pebble consumption rose to 1500 lb per day, 30.9 rpm, and 1650 lb on higher speeds, 34 rpm. When the load was changed over to a low discharge mill and raised from 0.45 load to 0.50 load, the consumption rose to 2400 lb per day at 30.9 rpm.

Pebble consumption varies with the size of rock feed; the larger the rock the greater the consumption. There is apparently some breakage of large pieces, and the resulting load is not proportionately larger on the top meshes.

Fig. 4—Pinion drive of new 6 ft 8 in. x 16 in. pebble mill. Note the shell clearance and corner cut off foundation to allow liner bolts to clear.

Conversion Plans

Mechanical Problems: The laboratory and the plant low discharge tests both indicated that satisfactory grinding could be done with pebbles, but the problem of converting the whole plant to pebble mills still remained. The problems were:

1—If the 5x16-ft mills were used, 20 would be required for a 1000-ton plant. All the motors would be running low power factor under the condition of half load. The Ontario Hydro Power Commission was already imposing restrictions on power.

2—Building new and larger mills would involve new motors, which were difficult to obtain, and an acute steel shortage meant that complete new mills were not available.

The problem was solved at little capital cost and with little difficulty in conversion by expanding the diameter of the shell of the mills until the new mill with pebbles would draw the same horsepower as a 5x16-ft high discharge steel mill, i.e., the normal capacity of the motors 180 to 185 hp. The original mills had large spur gears and would allow for the

required 10-in. expansion, only necessitating the chipping of the corner of the cement foundation under the pinion gear, see Figs. 3 and 4.

The expanded mills had to be slowed down in speed, requiring a 17-tooth pinion in place of the original 19 and 21-tooth pinions. A ⅝-in. welded steel shell was made to fit over the existing ends of the 5x16-ft mills, expanding them to 6 ft 8 in. in diam. See Figs. 3 and 5.

Allis-Chalmers engineers, particularly Mr. Hollefriend, together with Mr. Purdie, Lake Shore Mechanical Superintendent, helped to work out the final details of this relatively simple conversion plan. The ultimate cost of the conversion job was:

New 6 ft, 8-in. shell	$4,450.00
New pinion	400.00
Removing old shell, installing new shell, and changing pinion	200.00
Total cost per mill changed	$5,050.00

This mill could then be relined with a Ni-hard liner shell and feed end for $2,800, plus $400 for discharge grates.

In calculating the required diameter of a low discharge pebble mill to equal a high discharge 5x16 ft steel mill: 1—The capacity of 5x16 ft high discharge mill, 182 hp, is taken as 1.00. 2—The tests showed that a 5x16 ft high discharge pebble mill with lifter liners had a capacity of 0.34. 3—A 5x16 ft grate discharge pebble mill with lifter bars had a capacity 40 pct greater, or 1.4x0.34 = 0.475.

Then, let X = diameter of a grate discharge pebble mill with a capacity equal to a 5x16 ft high discharge mill = 1.0, and knowing that the capacities of mills vary as $d^{2.6}$

$$\text{Then } 0.475\,X^{2.6} = 5^{2.6} = \frac{65.6}{0.475} = 138.1.$$

Solving for X = 6 ft 7¾ in., use 6 ft 8 in.

Since in all tests with pebbles the horsepower per ton ground was directly proportioned to the capacity of the mill, the assumption was made that the 6 ft 8-in. would draw the same horsepower as the 5x16 ft high discharge mill, 182 hp. In the previous grinding report it had also been found that the capacity, power, and steel consumption varied as the 2.6 power of the diameter. This assumption proved to be correct.

Rock Feeding

The problem of handling 35 to 40 tons of rock per day (3½ to 4 tons per day to each of ten mills) without excessive labor charges was still to be solved. The original 5x16-ft mill was kept in pebbles by building a small hopper of 2½-ton capacity and filling it each day with the mill fork lift truck with a scoop shovel attachment. The pebble consumption is great enough that a pebble mill should be fed at least every 12 hr and preferably every 8 hr or even every 6 hr. Since this is a grate discharge mill, the load must be kept within 1 in. ± of the center line to get maximum capacity from the mill.[7]

The original 5x16-ft mills all had drum feeders. These would not handle 3-in. screened rock at a greater rate than 200 lb per min. Many of them were worn out, so it was decided to replace them with hopper feeders. The original trunnion liners were 7½-in. outside to 11 in. on the inside of the liner and would not take the elbow from an 8-in. pipe. It had been established previously that 3-in. rock required at least 8-in. pipe to carry it. These old trunnion liners were removed, and the ends of the mill were lined with ½-in. Linatex. These were built up to give a slope of 1 3/16 in. in 22 in. with an outside opening of 10 in. and an inside opening of 12⅜ in. This gave an opening big enough to allow a hopper with an 8 in. elbow at the bottom, see Fig. 6.

Fig. 5—New welded plate shells for 6 ft 8 in. x 16 in. pebble mills. The studs on the end fit over the original 5 ft tube mill ends.

These hoppers work well and allow the pebbles to be fed from an overhead conveyor running the length of the pebble mill floor with angle ploughs scraping the rock into the desired mill.

The Linatex lined trunnions have another operating advantage. It is not possible for grinding material to get behind the trunnion liner and wear out the trunnion itself without being noticed. The rubber is glued directly to the mill trunnion and wear must come from the outside where it can be seen.

There is also a power saving through using hopper feeders over the old drum type feeders. Our tests show that the drum feeders formerly used on the mills required about 5 hp to operate the feeder.

No. 8 Mill:	Maximum hp before removing drum feeder	190
	Maximum hp after removing drum feeder	185
	Saving	5
No. 7 Mill:	Hp immediately before removing drum feeder	158½
	Hp immediately after installing new hopper feeder and on the same pebble load (which was 4 in. low)	150
	Saving	8½

The problem of holding the pebble load within 1 in. of the center line was a difficult one. Two alternatives were considered:

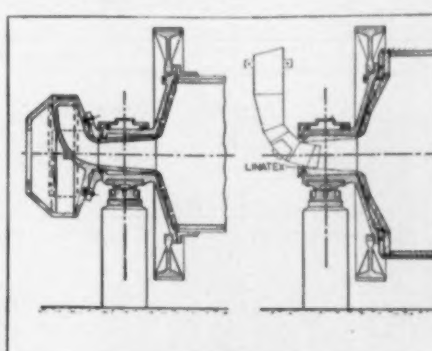

Fig. 6—Old arrangement, left, of 5x16-ft tube mill with drum feeder. New arrangement, right, of 6 ft 8 in. x 16 ft pebble mill with hopper feeder.

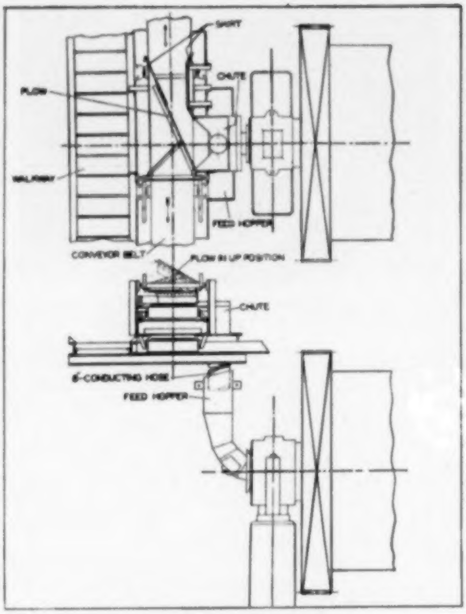

Fig. 7—Pebble feed system to pebble mill.

1—To put sensitive kilowatt meters on each mill so that the mill would be held at maximum power at all times and loaded each shift until the power reached its peak, since it has been established that the mill will draw maximum power when the load is exactly at the center line. The difficulty with this operation is that the power falls off when the load is both over and under the center point. Therefore it would be necessary to have a recording chart on each mill to obtain enough evidence to state definitely that a low horsepower meant that the load was low rather than too high. All types of recording kilowatt meters were investigated, and it was found that for ten mills the cost of any suitable installation of recording meters would be prohibitively high.

2—The other alternative was accurately to weigh the pebbles into the mill each shift according to their liner age, liner design, and type of grind (i.e., primary, secondary or tertiary) and to assume that the ore hardness would be consistent enough to maintain our load requirements. It has been found that Lake Shore ore, because of the sequence mining system employed underground to control rock bursting, was very consistent in hardness; also, 10 years' experience in having one man weigh steel into the mill had proved that accurate weighing methods were the most satisfactory.[8] These weighings would be backed up by kilowatt meter readings, of course, but on a relatively cheap meter such as a kilowatt demand meter. The Lincoln maximum demand meter type WD3 made by Sangamo has been found to be very suitable. The operators are warned not to read them within 20 min of a shutdown or power failure.

Fig. 8—Batch-weighing hopper.

This second alternative was the one finally decided upon, and it has proved most satisfactory.

Rock Screening and Handling

Preliminary screening tests indicated that Lake Shore ore screened through a 3x3-in. opening and retained on a 2x2-in. opening gave a pebble load in the mills with 51 pct of the pebbles −1½ in. and +¾ in. Tests showed that this load ground with about the same efficiency as a 1¼-in. ball load.

By varying the screens, pebbles of different sizes can be obtained easily.

Since the primary crushers are usually set at 1 in., this meant screening the ore as it comes up the shaft from the underground crushers (−6 in.).

This involved alterations to the crushing plant. It was decided to send all −¾-in. material in the hoisted ore (amounting to 30 to 35 pct) directly to

Table X. Shell Liner Costs

Type	Grinding Media	Type Liner	Material	Thickness Liner, In.	Total Weight Shell Liner, lb	Cost per lb. Nihard, c	Total Cost, $	Life, Days	Cost per Day, $	Cost[a] per Ton, $
5x16 ft H. D.	1¼ in. balls	M. Black Groove	Nihard	2-1/16	20,000	11.8	2360.00	600	3.93	0.032
6 ft 8 in. x 16 ft L. D.	1¾ in. pebbles	Hendry Wave	Nihard	1-11/16	20,000	11.8	2360.00	700	3.37	0.027

[a] Based on one rod mill, one ball mill and eight pebble or tube mills for 1000 tons per day.

Table XI. Comparison of the Initial Cost of Buying New Tube Mills and New Pebble Mills of Equal Grinding Capacity and Power

High Discharge Tube Mill	Equivalent Grate Discharge Pebble Mill	Ball Load, Tons	Rock Load, Tons	Cost of Initial Load Balls at $122.00[a] Per Ton, $	Rock	Complete Cost of New Mill Plus Nihard Liners Less Motor, $[b]		Tube Mill Plus Initial Load, $
						Tube Mill	Pebble Mill	
5x16 ft	6 ft 8 in. x 16 ft	18.0	14.0	2,210	—	17,300	19,770	19,510
6x16 ft	8 ft 0 in. x 16 ft	25.8	20.1	3,170	—	21,225	24,652	24,395
7x16 ft	9 ft 4 in. x 16 ft	36.2	27.4	4,450	—	30,150	33,995	34,600
8x16 ft	10 ft 8 in. x 16 ft	46.1	35.7	5,660	—	38,500	43,900	44,160

[a] January 1951 price.
[b] These figures are supplied by Canadian Allis-Chalmers Co.

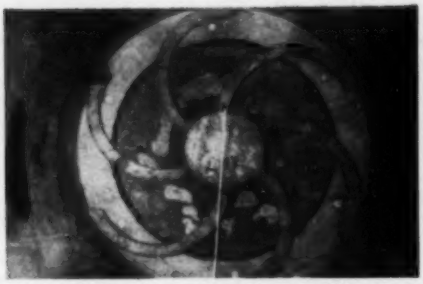

Fig. 9—Low discharge lifter scoops used on all pebble mills. The grate sections are placed directly at the front of these scoops resting on the short lugs shown in the center and flush with the center plate.

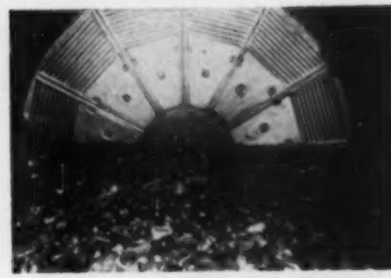

Fig. 10—Pebble load at discharge end. Note pebble load at center line and 5/16-in. slots in Nihard grate sections.

the rod mills. The +¾-in. material then was crushed in two open circuit Symons cones with a screen between them. This replaced a three cone and closed circuit screening plant. The removal of the −¾-in. material from the hoisted ore required a triple decked screen, the upper decks to protect and screen out the coarser material. Therefore, these decks were used, when necessary, to produce the correct size of rock feed for the pebble milling. Two triple decked 5x10-ft Ty-Rock positive circle throw screens were installed.

The rock that passes the top deck, 3x3 in., and is retained on the second deck, 2x2 in., then is conveyed to a single deck 3x6-ft screen on the top of the rock feed* storage bin of 300-ton capacity in the mill. This screen splits the rock feed into coarse and fine fractions, which are stored in separate halves of the main bin. Separate Jeffrey feeders draw either coarse or fine rock feed or a mixture for the pebble mills. These feeders discharge into the weighing hopper of the Toledo scales system. A batch of a definite weight up to 1000 lb can be withdrawn as needed, usually 200-lb batches at a time. These weighed batches are elevated with a bucket conveyor that discharges onto the conveyor belt running the length of the pebble mill floor and is located directly over the feed hoppers of the pebble mills. Ploughs have been installed opposite each mill and the rock feed directed to the desired mill through an 8-in. rubber hose. Classifier sands help to wash the pebbles through the linatex lined trunnion into the mill. See Figs. 6 to 11.

* The screened rock used to make pebbles is referred to as *Rock Feed*. The term *pebble* is not used until the rock feed has been in the mill long enough to become rounded.

On the 6 ft 8-in. mills, the primaries are fed 7900 lb per day of coarse rock feed, with the average theoretical mean diam 1.87 in. The final stages are fed 5300 lb per day of fine rock feed, with the average theoretical mean diam 1.44 in. The wattmeters are read every 2 hr and used to check the pebble load in the mill.

Pebble Mill Costs

Among the factors involved in pebble mill costs are the following:

1—The saving effected by using screened ore in place of 1¼-in. grinding balls has already been discussed. This saving amounted to $0.285 per ton milled at September 1951 steel prices.

2—To mill 1000 tons per day, one rod mill, one ball mill, and eight pebble mills were required. Each mill uses approximately 3½ tons of rock per day, 840 tons per month. This rock is taken from the underground ore before crushing. To crush and grind this rock to the fineness at which it leaves the pebble mills would cost $0.50 per ton. On 2.8 pct of the tonnage this amounts to $0.014 per ton milled. This saving should pay for running the extra conveyors and pebble feeding equipment.

3—The horsepower per ton milled is the same for pebble mills as for steel mills.

4—Liner costs are slightly cheaper in pebble mills. This saving, however, is only $0.005 per ton milled. Shell liner costs are given in Table X.

The 6 ft 8-in. pebble mills are easier to reline than 5-ft steel mills. By cutting the pebble feed for 1 day the load drops 7 to 8 in., and the mill can be relined easily without dumping the remaining pebble load. The worn liners are not so rigidly wedged together as they are with steel. The 5-ft steel mills were always relined by dumping the 18-ton ball load on the floor and then replacing it.

5—Operating labor is the same.

Fig. 11—Feed and liner arrangement. The eight outside 9-in. segments are not bolted.

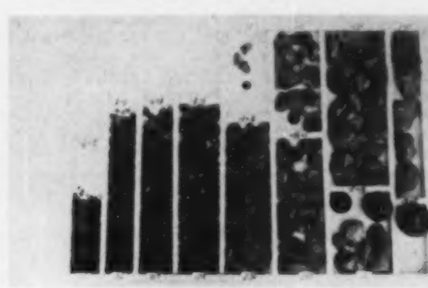

Fig. 12—Rock feed and pebble load tray.

Table XII. Sizing of Rock Feed and Resultant Pebble Load

Size, In.	Wt, g	Rock Feed March 11-17-20, 1950 Avg Wt per Piece = 117 G Pct Wt	Resulting Pebble Load March 1950
−2½ + 2	−356 + 184	29	Pct Wt
−2 + 1½	−184 + 78	55	14
−1½ + 1	− 78 + 23	15	28
−1 + ¾	− 23 + 9.7	1	23
−¾ + ½	− 9.7 + 2.9		17
−½ + ⅜	− 2.9 + 1.2		11
−⅜ + ¼			5
−¼ + 3/16			2
		100	100

6—Maintenance is slightly higher since all the mills have grate discharges.

7—To feed the pebbles takes 9 man hr per day at the moment as against 6 man hr per day for feeding steel. It is anticipated that this will be reduced to 6 hr as the plant is tuned up.

A comparison of the initial cost of buying new tube mills and new pebble mills of equal grinding capacity and power is given in Table XI.

It will be noted that although the large diameter pebble mill costs more than the tube mill to purchase, by the time the tube mill is loaded with balls at $122.80 per ton, the cost of the two installations are nearly equal.

Rock Feed and Pebble Load

A typical analysis of the primary rock feed and pebble load first used at Lake Shore is given in Table XII.

The rock feed was screened on a 3x5-ft Dillon screen (a temporary setup) through 2½ in. and +2 in. The pebble load was sampled by driving a 10-in. pipe down through the load at intervals.

The sizing of these odd shaped particles presented a problem, as ordinary screening on Tyler screens was unsatisfactory. It was found necessary to size by first roughly screening or sorting and then actually weighing each piece of rock between upper and lower limits to determine its proper size.

A rock feed of 5000 g and pebbles of 5000 g were sized by screening and weighing and are shown in Fig. 12. Steel balls marked with their sizes are shown for comparison.

The best guide for proper pebble size is to start by using a pebble of the same weight as the correct size of steel ball. Later, if the tonnage being handled is not too high (i.e., the time of contact in the grinding mill is relatively long) the pebble size may be gradually reduced to approach, but never quite reach, the same diameter as the steel ball. The smaller the grinding media, the greater the production of finished material, provided the top sizes are not allowed to build up.

Summary and Conclusions

1—Shape of grinding media is not important for −8 mesh grinding.

2—The capacity of the grinding media is directly proportioned to its specific gravity; thus siliceous ore pebbles, steel balls, and tungsten carbide balls are in the ratio of 0.34, 1.00, and 1.68.

3—A method of using a 12-in. grinding mill to evaluate plant fine grinding practice is explained.

4—High discharge pebble mills are erratic in performance.

5—Low discharge pebble mills use 40 pct more power and have 40 pct more capacity than high discharge pebble mills. Under identical conditions steel mills show only 27 pct difference.

6—A cheap and practical method of converting existing steel mills to pebble mills is outlined. A 5x16-ft mill is converted for $5050. Low discharge pebble mills have 47.5 pct of the capacity of the same mill filled with steel balls and equipped with standard trunnion discharge. Thus, to use the same motor and get the same capacity out of pebble mills, the capacity of the mill has to be increased 2.1 times. This can be done by 1—increasing the diameter on $d^{2.6}$ basis or 2—by lengthening the mill 2.1 times. The following table shows the expansion necessary on standard diameter mills for pebbles at 2.6 sp. gr.

H.D. Steel Mill Diam, Ft	Equivalent L.D. Pebble Mill Diam
5	6 ft 8 in.
6	8 ft 0 in.
7	9 ft 4 in.
8	10 ft 8 in.

The initial cost of each of these two mills, when loaded, is nearly the same.

7—The horsepower per ton milled is the same for pebble mills as for steel mills.

8—A method of feeding rock to the mills and of controlling the pebble load is described.

9—Liners with some lift—at least 1 in. in every second row—are required for pebble mills. Operating speeds are normal, 83 pct critical; but high speeds are more attractive than they are with steel balls.

10—Slightly more dilute pulps (7 pct less solids) are recommended.

11—For 1¾-in. and 1¼-in. pebbles, grates with slots up to 1/4 in. are best and 5/16 in. is maximum.

12—On Lake Shore ore the pebble consumption per 6 ft 8-in. mill varied from 5300 lb per day for 1¼-in. pebbles to 7900 for 1¾-in. pebbles.

13—Practice has been to start by using a pebble of the same weight as the steel ball being replaced.

14—The annual savings due to the installation of pebble milling amount to 785 tons of steel, or over $104,400.00. The saving amounts to $0.285 per ton milled.

Acknowledgment

The author would like to acknowledge the invaluable assistance of the following men who contributed to the success of this project: J. E. Williamson, formerly chief of the Lake Shore Research Staff, now with Union Corp. of South Africa, conducted the original experimental work. He was assisted in this by G. H. McCrank and Austin Murphy, C. Whitman, Mill Foreman, and the mill crew who helped to develop the operating end. J. G. W. Purdie, Mechanical Superintendent, G. Honer, Chief Electrician, and E. H. Bronson, Consultant, helped with the design and installation of the plant. J. C. Adamson, Mine Superintendent, and A. L. Blomfield, President, gave valuable advice and encouragement.

References

[1] Fine Grinding at Lake Shore. The Staff, Canadian Institute of Mining and Metallurgy. *Trans.* (1940) **43**, pp. 299-434.

[2] Addendum: Fine Grinding at Lake Shore. The Staff, Canadian Institute of Mining and Metallurgy. *Trans.* (1941) **44**, pp. 379-395.

[3] F. C. Bond: Standard Grindability Tests Tabulated. *Trans.* AIME (1949) **183**, p. 313; *Mining Technology* (July 1947), p. 2180.

[4] P. 409 ref. 1.

[5] B. S. Crocker discussion on paper by E. J. Roberts: The Probability Theory of Wet Ball Mining and Its Application. *Trans.* AIME (1951) **190**, p. 988; MINING ENGINEERING (November 1951).

[6] P. 387 ref. 1.

[7] P. 383 ref. 1.

[8] P. 396 ref. 1.

aime NEWS

Reinartz To Head Slate Of 1953 Nominations

The Nominating Committee for Institute Officers in 1953, John R. Suman, chairman, completed its work at the Annual Meeting in February, and has made its report to the Board of Directors. The President-Elect in 1953 serves as President in 1954. The ticket follows: For President-Elect and Director, Leo F. Reinartz; for Vice-President and Director, A. B. Kinzel and Lloyd E. Elkins; for Director, Hjalmar W. Johnson, Philip Kraft, Gail Moulton, Edwin L. Oliver, Philip Wilson, and Elmer Isern.

Mr. Reinartz is a vice-president of the Armco Steel Corp., Middletown, Ohio.

As required by the bylaws, brief biographical sketches of the nominees will be published in the July issues of the journals.

Industrial Minerals Div. To Hold Chicago Meeting

The annual fall regional meeting of the Industrial Minerals Div. has been scheduled for Sept. 3 to 6, 1952 in Chicago in conjunction with the Fall General Meeting of AIME. The program tentatively calls for a field trip on September 3 to the Illinois Brick Co., Blue Island. This is probably the largest brick plant in the Middle West. Another trip planned for the same day will be to the Indiana molding sands at Dune Sand State Park. These trips are under the co-leadership of the Illinois Geological Survey and Indiana Geological Survey. On Thursday, Sept. 4 there will be a tour of the open and underground mining of silica sand, washing, and processing, at the Standard Silica Corp. Open-pit mining at Illinois Clay Products Co., Goose Lake, will also be visited on Thursday. Transportation for the tours will be by chartered bus.

Technical sessions will be held on September 5 and 6. A survey of industrial minerals locally produced or imported for use in the Chicago area and a session on building materials is planned for Friday. Industrial groundwater will be discussed on Saturday, featuring a description of the methods used in exploration for developing and production of groundwater for industry.

The division is considering the possibilities of joint participation with other groups and will also sponsor at least one social function at Chicago.

Robert C. Stephenson is chairman of the fall meeting committee.

Deadline Set for 1953 Annual Meeting Papers

Sept. 15, 1952 is the deadline for receipt at Institute headquarters of papers for the 1953 Los Angeles Annual Meeting that may be prepublished. Papers may be listed in the program at the discretion of the Divisional program committee chairmen up to such time as the chairmen have completed their respective programs. It is suggested that authors who do not make the Sept. 15 date, submit their papers as soon thereafter as possible rather than to wait until after the Annual Meeting. Papers are scheduled for publication in the approximate order of acceptance. These copies, each complete with illustrations, are required form.

Surplus Mexico City Programs Are Available

The surplus supply of the official programs of the Mexico City meeting, Oct. 29 to Nov. 3, 1951, has been sent to New York headquarters and they are available on request. The program contains abstracts of the papers presented.

In the story of the Mexico City meeting, no mention was made of the very fine cocktail party and buffet given on Sunday, Oct. 28. The party for all meeting registrants, signaled the opening of the meeting and was given by the Colorado Fuel & Iron Corp.

AIME Coal Div. Plans Joint Meeting with ASME

The next scheduled meeting for the Coal Div., AIME, is the joint meeting with the Fuels Div., ASME, Bellevue Stratford Hotel, Philadelphia, Oct. 30 to 31, 1952. D. C. Helms, Lehigh Navigation Coal Co., Lansford, Pa., is the AIME Coal Div. Co-Chairman for this meeting.

The annual banquet of the School of Mineral Industries at the Pennsylvania State College was held on Mar. 22, 1952. A highlight of the occasion was the presentation of a scroll to John J. Forbes, director of the U. S. Bureau of Mines. The testimonial stated "... in recognition of his 40 years of distinguished service to the cause of mine health and safety which culminated in his appointment to the Directorship, United States Bureau of Mines, November 6, 1951 ..." Mr. Forbes was a member of the class of 1911. Left to right: Millard Rehburg, Dean Edward Steidle, John J. Forbes, Melvin Ott.

> **Classification and Thickening:** Cyclones are attracting wide interest in this field. Their use as primary thickeners in coal preparation plants is reported to require lower investment costs and less space over conventional thickeners. They are claimed to deslime effectively down to 10 microns, or to thicken with very dense spigot products......

Excerpt from the article "Mineral Dressing" which appeared in the February, 1952 issue of Mining Congress Journal.

MILLING and ORE DRESSING MEN will be interested in H&P CYCLONE THICKENER EQUIPMENT

... as THE ANSWER to Many of Their Classification & Thickening Problems

The Heyl & Patterson Cylone Thickener is a device utilizing Centrifugal Force to separate a suspension of solids in water into a thickened underflow and a low concentration overflow. The H&P Cyclone Thickener generates Centrifugal Forces up to 12,000 times gravity to separate the solids from the fluid.

Hundreds of Heyl & Patterson Cyclone Thickeners are now in use in the mining industry . . . controlling solids in Circulating Systems and separating solids from liquids.

The Cyclone equipment provides an economical and practical method of clearing up problems of:

PULP Thickening . . .
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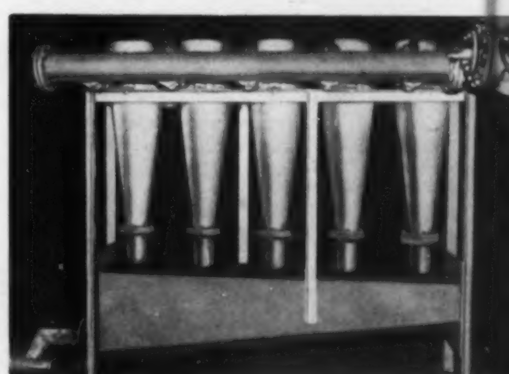
Heyl & Patterson laboratory and field tests have provided a sound basis for predicting accurately the Cyclone performance on most feeds. For further information ask for Booklet 5-CT-51 or send us your problem and we will tell you how a Cyclone application can provide the answer.

ADVANTAGES OF CYCLONE THICKENER EQUIPMENT

1. Occupies a small amount of space. 2. Has a low initial cost (only a fraction of other thickening methods). 3. Needs a minimum amount of upkeep. 4. No moving parts.

The standard 3" Cyclone Manifold consisting of a feed valve, combination feed chamber and overflow chamber, overflow valve, underflow pan and twenty-two Cyclones. The standard 3" Manifold consisting of twenty-two Cyclones has a capacity of 250 G.P.M. and is designed to operate at 40 PSI.



A typical manifold of five 14" Cyclones. These larger Cyclones are usually operated on lower pressure and are generally used as Classifiers.

Cyclone Thickeners
Thermal Dryers
The Drying Dutchman
Reineveld Centrifugal Dryer
Thorsten Sampling Systems
Ore Bridges
Railroad Car Dumpers
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HEAVY BULK MATERIALS HANDLING EQUIPMENT

ALL THE WAY FROM DESIGN TO ERECTION

Industrial Minerals Div. Plans Nova Scotia Meeting

Long range plans are being made by the Industrial Minerals Div. for a meeting in Nova Scotia in August or early September 1953. The invitation for the meeting was received from the Nova Scotia Mining Society. The Industrial Minerals Div. of the Canadian Institute is also expected to participate.

Headquarters for the meeting will probably be the summer field study camp at Crystal Cliffs, 120 miles east of Halifax. Crystal Cliffs will be a good jumping off point for the numerous field trips which will be an important part of the trip.

The Executive Committee of the Division is beginning early on plans for the meeting so that travel arrangements and accommodations will be of the highest order.

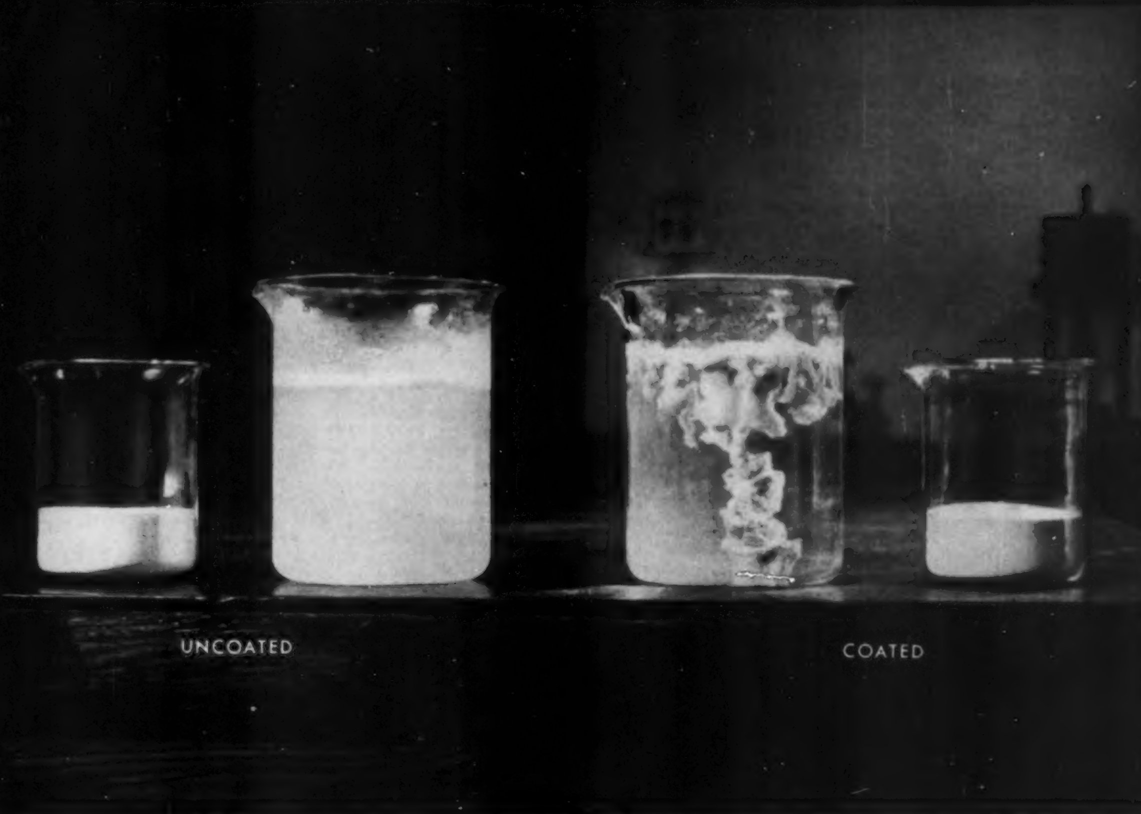
Return Cards Promptly For New Directory Listing

Last month, reply postal cards were sent to all AIME members asking for information for listing their names in the 1952 Directory, to be issued late in the year. Members are urged to return the cards as promptly as possible, with information as to their company connection and address. If cards are not returned the address to which the cards are mailed, and which is given in the upper right-hand corner of the reply card, will be used in the Directory. This is the address to which we are now sending AIME publications. In many instances it is the home address of members, whereas the address and position given in the Directory is customarily the company or office address. All records are being transferred to the Flexoprint system, which takes some months to complete. When completed, a Directory can be printed on short notice at any time. The geographical section of the Directory will revert to the straight geographical arrangement, by countries, states and provinces, and towns, as used in the 1948 and preceding Directories. The geographical listing of members' names will include for the first time the major Divisional interest of each member as well as the Divisions in which he is enrolled, which information is on stencil but not on return card.

If you have not returned the card, do so now. Advise headquarters immediately if you change your position or address, giving both the address you wish used for publications and the listing you wish in the Directory.

Copies of Mining Engineering for February and March, 1952 are needed at Institute Headquarters in New York. The Institute will pay $0.50 per copy for these issues when returned.

PROGRESS IN EXPLOSIVES . . .



WHICH BEAKER HAD THE BANG?

The small beaker (left) contains ammonium nitrate. This is an important explosives ingredient, but it absorbs moisture readily. Notice in the large beaker (left)—how it dissolves immediately in water.

The small beaker (right) contains *Hercules* ammonium nitrate *treated with a special resin*. In the large beaker (right) note how the resin-treated ammonium nitrate repels water without affecting explosives' properties.

Here is a simple example of Hercules' pioneering that gives you better blasting efficiency under adverse storage and climatic conditions both in manufacture and in field use.

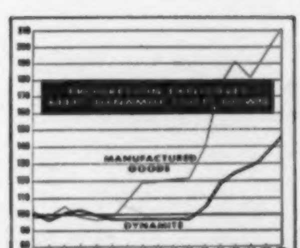


Chart shows relative stability of dynamite prices since 1935, as compared with prices of other manufactured goods. 1935-39 values=100.

HERCULES POWDER COMPANY

INCORPORATED

Explosives Department, 955 King St., Wilmington 99, Del.

XR52-4

THE DRIFT OF THINGS

by Edward H. Robie

NEVER before have the annual company reports in the mineral industry field exhibited the typographical art so abundantly as does the current crop. Time was when most company reports made a drab appearance and dry reading, mostly confined to financial data and text that gave little information to the public or technical men. Now, the dollar statistics assume less importance. Considerable information is given on what is going on at the company's plants, new equipment being installed, new methods, properties under development, ore reserves and grades, and similar information of interest to professional men working in the field. We have one suggestion for further improvement: The names of the officers and directors of the company are always given, but only a few companies give the names of the principal operating officials. We think, in the case of a mine for example, that the name of the general manager, his assistant, the mine and mill superintendents, and any other top operating officials who have been largely responsible for the success of operations, are worthy of mention. The reports are now widely read and this recognition is due them.

Conspicuous among the annual reports that we happen to have seen, in point of typographical attractiveness, are that of the Texas Co., celebrating its fiftieth anniversary with a gold-plated cover and gold printing inside; and that of Noranda Mines Ltd., with a red and black cover that would attract attention anywhere.

Comment on economics, taxes, and labor give an excellent picture of what our industry is facing. Consider the following, from the report of the Union Oil Co. of California:

"Your Company, as well as industry generally, is finding it extremely difficult to maintain the financial resources necessary to cover the high cost of replacing raw material reserves and facilities and at the same time expand its operations. Not only are taxes severely limiting the accumulation of capital by industry, but they are reducing the savings of individuals, the basic source of risk capital, and eliminating the incentives for investment. Simultaneously industry is prevented by price control from recovering most increases in cost brought about by inflationary wage increases and taxes.

"As a result of the inflation induced by unbridled Government spending, we can no longer regard profit as simply the difference between sales revenues and recorded costs. The reason is that only a portion of recorded costs are expressed in today's dollars. Depletion and depreciation costs which provide for the recovery of capital invested in properties and facilities are limited to the number of dollars initially invested. Because of the decline in the purchasing power of today's dollar, the amounts set aside for depletion and depreciation are simply not enough to meet replacement costs. Therefore, it is a literal fact that a substantial portion of the amount reported as profits is actually not profit, but should be recorded as a return of capital needed to keep the Company a going concern. It is not too much to say that this portion of apparent profits on which your Company is taxed, represents an involuntary liquidation of capital assets. There is a serious doubt whether your Company's reported profit figures, in common with most of industry, have any real meaning . . .

"It is seldom recognized that the purchasing power of industry's dollar has declined even more than has that of the individual. During the past 10 years, the wholesale price index, a measure of industry's cost of doing business, has increased 107% while the consumer's price index has increased 76%."

A Practice That Should Spread

An idea materialized at the recent Annual Meeting that we should like to see extended. Whether held in New York, San Francisco, Los Angeles, Chicago, or St. Louis, a comparatively small proportion of the Student Associates and Junior Members of the Institute are able to attend its Annual Meetings. They can gain a great deal in technical knowledge, in widening their acquaintance, and in absorbing the spirit of a profes-

sional society in action by coming to such a meeting, but must perforce be absent because of either the cost or the requirement of their employer or their school that they should remain on the job.

The problem was completely solved for 37 young men last February. A picked group of that number from the School of Mines and Metallurgy at Rolla, Mo., was excused from classes for a week and organized by J. D. Forrester, Chairman of the dept. of mining engineering. The Sinclair Coal Co., Kansas City, Mo., through the good offices of Thomas C. Cheasley, footed the bill of $1300 for a bus from Rolla to New York and back. Free registration at the meeting was provided by the Institute, and hotel expense was kept nominal by lodging most of the group at Sloan House, the neighboring YMCA, at $1.35 per person per day. We suspect that the AIME Directory ten years hence will show practically every one of these young men as a member.

Many other companies would doubtless do something similar if the head of the department at a college would promote the idea. Let us have more such trips to the Los Angeles meeting next February.

Junior Members cannot be similarly grouped for attendance at an Annual Meeting, but their employers can offer such a trip, with a part, at least, of expenses paid, as a prize for conspicuously satisfactory work. From the company standpoint the money would be well invested on several counts. If you, who read this, happen to be the head of a company or local operation, why not announce, now, that one or more of your Junior Members will have the opportunity to go to the Los Angeles meeting? Alert Junior Members will doubtless find a way of bringing this suggestion to the attention of their boss!

A Complete Library

The statement has been made, and we believe never questioned, that the AIME Transactions constitute "the most extensive and authoritative library extant on engineering and technology in the mineral industry." Carl A. Zapffe evidently thinks so, for headquarters recently helped him complete, so far as possible, his library of the Transactions. He recently wrote, "Thanks again for the numerous favors connected with this operation. You can get one assurance from it, and that is that at least one AIME member will have paged through and studied every single volume of the Transactions since 1871! I expect to know everything that the Institute has done before these babies take their place on the shelves."

Dr. Zapffe is a metallurgist and according to the last Directory is available for consulting work. A word to the wise is sufficient!

Personals

J. R. Aker is now design engineer with Black, Sivalls & Bryson, Inc., Oklahoma City, Okla.

B. Frank Allison, senior in the University of Kentucky College of Engineering, has been named winner of the Old Timers' award. This is an annual presentation to the school's outstanding mining engineering student.

J. Donald Allan, chief geologist of the Province of Manitoba, Mines Branch, has left the government and is now geologist with the California Standard Co., Calgary, Alberta.

ROBERT T. BANKS, JR.

Robert T. Banks, Jr., consulting engineer, is in Belgrade, Yugoslavia as M.S.A. consultant to the Yugoslav Government on mining and metallurgical industries.

David W. Bushell has joined the Britannia Mining & Smelting Co., Britannia Beach, B. C. He had been with the Mufulira Copper Mines Ltd., Northern Rhodesia.

R. C. Bacon has resigned from W. R. Grace & Co. to join the staff of Panaminas Inc., New York.

Walter F. Clarke, assistant general manager, Independent Coal & Coke Co., has been elected president of the Utah Coal Operators' Assn.

Patrick Denis Coakley is now mill superintendent at the Empresa Minera Bolsa Negra, La Paz, Bolivia.

John B. Calkins, formerly with the basic industries research laboratory, Allis-Chalmers Mfg. Co., Milwaukee, has joined the Zonolite Co., Libby, Mont.

Douglas C. Corner is with John Carruthers & Co., Sydney, Australia.

Garnet G. Copeland has resigned from the Fluosolids Div., Dorr Co. to become vice-president and general manager of the Dominion Silica Corp., Ltd., Montreal.

Donovan A. Dutton is now with the W. P. Wooldridge Co., Los Angeles, as sales engineer. He had been associated with the Elkins Machine & Mfg. Co., Hobbs, N. Mex.

Wallace G. Fetzer has joined Oglebay Norton & Co., Duluth, Minn.

Barney Foust has become mill superintendent, San Mauricio Mining Co., Camarines Norte, Philippines. He had held the same position with the Atok-Big Wedge Mining Co.

Bertram M. Fairlie is now employed as a safety engineer by the Canadian Johns-Manville Co., Ltd., Asbestos, Que.

Emilio Jimeno Gil, Ph.D., professor at the University of Madrid, Spain, was elected a full member of the Spanish Royal Academy of Sciences, Exact, Physical, and Natural.

OSCAR GLAESER

Oscar Glaeser formerly assistant to the general manager in charge of western operations for U. S. Smelting & Refining Co., was promoted to assistant general manager.

Robert Y. Grant, formerly chief, mining and geology div., Natural Resources Section, SCAP, Tokyo, has moved to Taiwan, Formosa, where he is now special assistant in the office of chief of mission, Mutual Security Agency, Mission to China.

Henry Liggett Hosmer has been made geologist, Morococha Div., Cerro de Pasco Corp. Mr. Hosmer had been assistant geologist.

Arthur Curtis Hall is managing director, Northern Peru Mining & Smelting Co., Lima, and also managing director of the Cia. American Smelting Boliviana, S.A., La Paz.

M. L. Haider, 1952 AIME President, was appointed deputy coordinator of the oil producing activities for Standard Oil Co., N. J.

L. S. Hayes, American Smelting & Refining Co., Tucson, has been transferred to the mining dept., New York.

Charles E. Heinrichs is vice-president, Virginia-Carolina Chemical Corp., Richmond, Va.

M. A. Kuryla, industrial relations engineer for U. S. Smelting, Refining & Mining Co., has been appointed industrial relations manager for the western Div., Salt Lake City.

Frederic L. Kadey, Jr. is now at the research center, Johns-Manville Corp., Manville, N. J.

Klass Koeten has resigned from the Liberia Mining Co., Ltd., Monrovia, Liberia, where he had been assistant general manager. He is now with the ore dept., Handels en Transport Maatschappij Vulcann, Rotterdam, as engineer in charge of mining.

Richard E. Lindberg, Joy Mfg. Co., has been transferred from El Paso to Claremont, N. H.

Esworthy P. Lange has joined C. Tennant Sons & Co., Inc., New York as mining geologist.

John D. McAuliffe is now general operating engineer, M. A. Hanna Co., Iron River, Mich.

Mat J. Meinen has joined the Universal Cyclops Steel Corp., Bridgeville, Pa.

W. D. MC MILLAN

W. D. McMillan, U. S. Bureau of Mines, Denver, is making a study of the production possibilities for manganese in western Arizona.

William George McEvilly accepted a position with the Atlas Powder Co., Chicago.

Personals, Continued

John M. Martin has been appointed assistant chief chemist in the eastern district for Oliver Iron Mining Div., U. S. Steel Co., Virginia, Minn. He was formerly located at Coleraine in the Canisteo district for the firm.

MIRO MIHELICH

Miro Mihelich has been appointed manager of Harbison-Walker Minerios, Ltd., a subsidiary of Harbison-Walker Refractories Co. organized for developing sources of raw materials in Brazil.

Lester G. Morrall has been made chief of the North American Div. of Region IX (foreign minerals) of the Bureau of Mines, Washington, D. C.

Louis E. Marchi, inorganic chemist in the ceramics and minerals research dept., Armour Research Foundation of Illinois Institute of Technology, has been promoted to senior chemist.

Gordon Francis McClay and **M. D. MacLennan** have been appointed managers of engineering purchasing, respectively, for the Canadian Ingersoll-Rand Co., Ltd., Sherbrooke, Que. **G. M. Dick,** former manager of engineering and purchasing, has been named chief engineer of the company.

William E. Mahin, director of research at Armour Research Foundation of Illinois Institute of Technology, has been named to the subcommittee on aircraft structural materials of the National Advisory Committee for Aeronautics.

Donald R. MacLaren is with the mineral deposits branch, U. S. Geological Survey, Denver.

W. R. McCormick, Jr. has recently been appointed superintendent of Calera Mining Co.'s cobalt refinery at Garfield, Utah.

Charles R. Nailler is now with the Christopher Coal Co., Pursglove, W. Va.

Bruce Netschert has completed his assignment with the President's Mate-

Personals, Continued

rials Policy Commission and has joined the staff of the Natural Resources Office of the National Security Resources Board.

Ernest O. Ohsol, General Electric Co.'s chemical div., has been appointed manager of the new product development laboratory.

CHARLES O. PERLWITZ

Charles O. Perlwitz, construction superintendent, has resigned from the Compania Minera Aguilar, S. A., Jujuy, Argentina, to accept a position with the Empresa Minera Santa Fe, Oruro, Boliva.

Obituaries

Sam S. Coldren (Member 1937) died while enroute to the February 1952 Annual Meeting. Born at Muscatine, Iowa on Sept. 23, 1900, Mr. Coldren was educated at Stanton Military Academy, Va.; New Mexico Military Institute; and the Colorado School of Mines. After graduation he was employed at various mines in Montana, Arizona, and New Mexico. In 1925 he joined the Cia. Minera de Clarines in Mexico as engineer in charge of examinations. He remained in Mexico for several years. Mr. Coldren went to the Philippines, where he was connected with the Nielson Enterprises, including the Lepanto Copper Co. He also operated a mine outside Hongkong. During World War II he was engaged in special work for the U. S. Government in Panama, Peru, and Bolivia. He returned to the Philippines in 1945 where he managed a manganese property in Mindanao for the Soriano interests. He recently had returned to the United States and was operating properties at Tucson, Ariz.

Nelson Dickerman (Member 1915) died on February 18. Mr. Dickerman was born at Denver in 1881 and graduated from the University of California in 1905 with the degree of B.S. In 1905 he joined the Yuba Consolidated Goldfields, Hammonton, Calif. In 1910 he became assistant to the general manager for Natomas Consolidated, San Francisco. For two years he was employed by the Kirtley Creek Gold Dredging Co., Salmon, Idaho, as superintendent and in 1913 Mr. Dickerman went to Colombia. In 1916 he returned to the United States and worked as a consulting engineer in California, Texas, Oregon, Nevada, and New Mexico. Several years later Mr. Dickerman was made vice-president and general manager for the Guiana Development Co. and Liberty Development Co. in Dutch and French Guiana. In 1924 he made examinations in Chili, Argentina, British Guiana, and the United States. He joined the Yuba Mfg. Co., San Francisco as engineer and in 1928 went to England for the company. Returning to California in 1932 he resided in San Francisco. Several years later he moved to Washington, D. C., where he was living at the time of his death.

John M. Moubray (Member 1906), a Life Member, died on Oct. 4, 1951. Born in Scotland in April 1881, he attended the Camborne School of Mines, Cornwall. In 1901 he joined the Berehaven Copper Mines, Ltd., Ireland, as assistant manager. A few years later he was in Rhodesia for the Remheim Gold Mines as surveyor and assayer. In 1904 Mr. Moubray joined

(Continued on page 520)

Obituaries

the Jumbo Gold Mining Co. and then returned to the Bernheim firm in 1905 as manager. Mr. Moubray was assistant engineer for the Bechuanaland Exploration Co., Ltd. At the time of his death he was residing at Shamva, Southern Rhodesia.

Ernest F. Pullen (Member 1949), president of the Alexo Coal Co., Oakville, Ont., has died. Mr. Pullen was born in Oakville in 1880 and attended the University of Toronto from 1901 to 1905. From 1906 to 1909 he was employed by the Transcontinental Railway Co. as an instrumentman. For three years he was resident engineer and in 1912 joined the Alexo Mining Co., Ltd. as manager. During the first world war he was in France and following his discharge returned to the Alexo Mining Co. In 1912 he was made general manager and president.

Joseph R. Linney

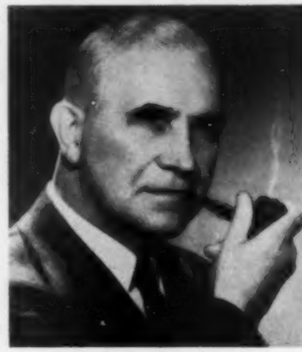
An Appreciation by Arthur T. Ward

Joseph Robert Linney died in Lima, Peru, on the morning of February 19. It was his intention, after visiting the leading capitals, to journey down the Amazon from Iquitos, Peru, and perhaps record his impressions for another book.

Joe Linney was a most extraordinary man. He was completely self-made, and his life story had in it much of that of a Horatio Alger hero. He was born in Old Forge, Pa., on May 7, 1888, of Welsh stock. He attended school only until the age of nine years, when he found employment in the nearby anthracite coal mines, as an "assistant to a mule"—in his own words.

By the time he was 21 years old, he had become a superintendent, and when 25 he became manager of a group of mines.

The Delaware & Hudson Co., at whose coal mines in Pennsylvania, Joe Linney was working, owned the iron mining operation at Lyon Mountain, N. Y., in the heart of the Adirondack country. The property needed competent management, and at the age of 31 Joe was placed in charge. This proved to be his great opportunity, and for the next 25 years he guided the destinies of this operation, which stands today the most important of its kind in the United States. On February 17, 1944, on the 25th anniversary of his arrival in the Adirondacks, Joe Linney retired as vice-president, and turned the active direction of the mining developments in that area (now a part of the Republic Steel Corp. operations) over to his sons and, with Mrs. Linney, retired to Plattsburgh, N. Y., to serve as a



JOSEPH R. LINNEY

mining consultant, and to pursue his many hobbies.

Joe Linney was singularly fortunate in his marriage to Elizabeth Davies, on Apr. 2, 1907, and in his family life. For the next 44 years, until Mrs. Linney's death, they comprised a perfect team, and together they planned and worked unselfishly for the welfare of the people of the community where they lived. The Linney's had two sons and a daughter. The elder, Robert J. Linney is now manager of operations, Reserve Mining Co., Babbitt, Minn., and William J. Linney, succeeded his father, at Lyon Mountain and Port Henry. Eleven grandchildren and two great grandchildren survive.

Joe Linney joined the American Institute of Mining and Metallurgical Engineers in 1920. He was a founder and last chairman of the Eastern Magnetite Mining and Milling Methods Committee, and always contributed much to the Annual Meetings.

He was always a leader in civic affairs, and was an officer or sat on the boards of directors of numerous groups, such as community hospitals, schools, churches, clubs. Governor Dewey had also appointed him a member of the Champlain Basin Commission.

To those who had the good fortune of being associated with Joe in any way, he was and will always remain, their hero.

NECROLOGY

Date Elected	Name	Date of Death
1922	Robert C. Coffey	January 1952
1941	N. L. Dorn	Mar. 2, 1952
1920	H. W. Gould	Nov. 14, 1951
1944	John G. Green	July 1951
1940	Thomas J. Hamilton	October 1951
1929	M. D. Harbaugh	Mar. 19, 1952
1920	Joseph R. Linney	Feb. 19, 1952
1920	Edward G. Mahin	Feb. 4, 1952
1923	Alan Morris	Mar. 18, 1952
1947	John G. Munson	Mar. 29, 1952
1936	Stuart W. Norton	Unknown
1905	Ambrose E. Ring	Feb. 29, 1952
1941	Ellsworth H. Shriver	Aug. 12, 1951
1948	Walter W. Stegman	July 30, 1951
1946	C. C. Van Deventer	May 18, 1951
1939	A. L. Weil	Unknown
1901	Charles Henry White	Mar. 17, 1952

Coming Events

Apr. 26-May 11, Liege International Fair, Liege, Belgium.

May 4-8, Electrochemical Society, Inc., Spring meeting, Benjamin Franklin Hotel, Philadelphia.

May 5-6, National Air Pollution Symposium, Huntington Hotel, Pasadena.

May 5-7, American Mining Congress, coal convention, Netherland Plaza Hotel, Cincinnati.

May 6-9, Scientific Apparatus Makers Assn., annual meeting, Edgewater Beach Hotel, Chicago.

May 9-10, AIME, joint meeting, Industrial Minerals Div. and Metals Branch, Davenport Hotel, Spokane, Wash.

May 11-14, American Institute of Chemical Engineers, Atlanta Biltmore Hotel, Atlanta.

May 17, AIME, Chicago Section, President's night dinner dance, Shoreland Hotel, Chicago.

May 27, AIME, Morenci Subsection, Longfellow Inn, Morenci, Ariz.

June 2-4, Chemical Institute of Canada, annual conference, Montreal.

June 4-14, Mechanical Handling Exhibition and Convention, Olympia, London.

June 6, AIME, Columbia Section, dinner meeting, Spokane Hotel, Spokane.

June 16-20, American Electroplaters Society, Industrial Finishing Exposition, International Amphitheater, Chicago.

June 23-27, ASTM, 50th anniversary meeting, Hotel Statler, New York.

July 1-Sept. 30, Centennial of Engineering, Chicago.

Sept. 3-6, AIME, Industrial Minerals Div., annual fall regional meeting, Chicago.

Sept. 3-6, AIME, fall meeting, Palmer House, Chicago.

Sept. 11-13, American Institute of Chemical Engineers, Palmer House, Chicago, Ill.

Sept. 22-25, American Mining Congress, Metal and Nonmetallic Mining Convention and Exposition, public auditorium, Denver.

Sept. 23-25, Institution of Mining and Metallurgy, symposium on mineral dressing, Royal School of Mines, London.

Sept. 30, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Oct. 3, AIME, National Open Hearth, Southern Ohio Section, Deshler-Wallick Hotel, Columbus, Ohio.

Oct. 20-22, AIME, Institute of Metals Div., fall meeting with National Metal Congress, Hotel Adelphia, Philadelphia.

Oct. 28, Assn. of Consulting Chemists and Chemical Engineers, Inc., annual symposium, Hotel Belmont Plaza, New York.

Nov. 4, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Nov. 6-8, New Mexico Mining Assn. and **International Mining Days**, joint convention, Alvarado Hotel, El Paso.

Nov. 19, American Mining Congress Coal Div. Conference, Wm. Penn Hotel, Pittsburgh.

Dec. 2, American Mining Congress, annual membership meeting, University Club, New York.

Dec. 4-6, AIME, Electric Steel Furnace Conference, Hotel William Penn, Pittsburgh.

Dec. 7-10, American Institute of Chemical Engineers, annual meeting, Hotels Cleveland and Carter, Cleveland.

Dec. 8, AIME, Arizona Section, all-day meeting, Tucson.

Feb. 16-19, 1953, AIME, annual meeting, Statler Hotel, Los Angeles.

Apr. 20-22, AIME, National Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Hotel Statler, Buffalo.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Feb. 29, 1952 was 17,221; in addition 2,341 Student Associates were enrolled.

ADMISSIONS COMMITTEE

T. D. Jones, Chairman; Thomas G. Moore, Vice-Chairman; H. S. Bell, F. W. Hanson, R. H. Chadwick, T. W. Nelson, C. A. R. Lambly, John T. Sherman, A. C. Brinker, G. P. Lutjen, Ivan Given, E. A. Prentis, C. Leslie Rice, Jr., and J. H. Scaff.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—Thiele, Elmer E. (J) (C/S—S-J)

Arizona
Ajo—Rickard, Forrester B. (M) (C/S—A-M)
Ajo—Sullivan, William T. (A) (C/S—S-A)
Miami—Kuck, David L. (J) (C/S—S-J)
Phoenix—Allen, Alden W. (J) (C/S—S-J)
Phoenix—Twitty, Howard A. (A)

Arkansas
Alexander—Thomas, H. Edwin (J) (C/S—S-J)

California
Los Angeles—Wright, Lauren A. (M) (C/S—A-M)
Pleasanton—Barneyback, Robert S. (M)
Ross—Livermore, John S. (M) (C/S—A-M)
Sacramento—Reynolds, Robert W. (J) (R. C/S—S-J)
San Francisco—Alexander, Roy K. (M) (R. C/S—J-M)
San Francisco—Simons, John P. (M) (R. C/S—J-M)
San Jose—Townsend, Richard P. (J) (C/S—S-J)
Whittier—King, Dan M. (J) (R. C/S—S-J)

Colorado
Climax—Hipp, Arthur W. (J)
Leadville—Lampert, John W. (J) (C/S—S-J)
Uravan—Carter, William D. (J)

District of Columbia
Washington—Ericksen, George E. (M)

Florida
Brewster—Albrecht, Theodore J. (A) (R. C/S—S-A)
Brewster—Chesney, James P. (J)

Georgia
Dallas—Phillips, Harris E. (J)
Savannah—Aikin, Wilbur O. (J) (R. C/S—S-J)

Illinois
Chicago—Zilbersher, William M. (J) (C/S—S-J)

Louisiana
Baton Rouge—Murray, Grover E. (M)

Massachusetts
Boston—Wheeler, Alfred H. (J) (C/S—S-J)
Cambridge—Smith, Edgar E. N. (J) (C/S—S-J)

Michigan
Ishpeming—Bjorne, Calvin C. (J) (R. C/S—S-J)
Negaunee—Kennedy, Allen D. (J) (C/S—S-J)

Minnesota
Duluth—Pearce, Clyde M. (M) (R. M)
Marble—Leach, Hugh J. (M) (C/S—A-M)

Montana
Butte—Jordan, Marcus C. (J) (C/S—S-J)

Nevada
Yerington—Goudey, Hatfield (M)

New Jersey
Dover—Casiraghi, Gilbert A. (J)

Appraisals
Assayers
Chemists
Construction
Consulting
Designing

Professional Services

Space limited to AIME members or to companies that have at least one member on their staffs. One inch, $40 per year; half inch, $25 payable in advance.

Mountain Lakes—Orsino, Joseph A. (M)
Plainfield—Pickard, Joseph W. (M) (R. C/S—J-M)

New Mexico
Carlsbad—Durland, Ross M. (M)
Carlsbad—Knill, Raymond R. (M) (R. M)
Carlsbad—Rasor, James H. (M)

New York
New Rochelle—Schmid, Walter P. (A)
New York—Atkinson, John R. (J) (C/S—Auto A-J)
New York—Hedges, Robert W. (A)

North Carolina
Kings Mountain—Rainey, Dudley L. (M) (C/S—S-M)

Ohio
Cleveland—Feucht, Jacob E. (M)
Parma Heights—Reinker, Norman W. (A)

Oklahoma
Picher—Burns, Otis A. (J)

Pennsylvania
Allentown—Rader, Herbert L. (J) (C/S—S-J)
Bethlehem—Dodson, John E. (M)
Camp Hill—Carrington, Winston H. (M) (R. C/S—J-M)
Dunmore—Kranick, Cyril R. (M)
Easton—Rich, Herbert W. (M) (R. C/S—J-M)
Lebanon—Ketter, Richard P. (J)
Lebanon—Moyer, William H., Jr. (J)
Schuylkill Haven—Morgan, Lawrence E. (M)

Tennessee
Nashville—Jewell, Willard B. (M)

Texas
El Paso—Smith, Richard S. (A)
Fort Worth—Jones, Victor H. (M)

Utah
Kenilworth—Jackson, Elias O. (M)
Salt Lake City—Alderman, Sidney S., Jr. (J)
Sunnyside—Dodds, Arnot W. (M)
Sunnyside—Hyatt, Vincent (M)

Virginia
Dante—Hughes, Robert H. (M)

Washington
Bremerton—Lampson, Richard J. J. (J) (C/S—S-J)
Seattle—Olive, Rosh M. (A)
Spokane—Naylor, Joseph Y. (J)

Wisconsin
Cedarburg—Molenhauer, Marvin I. (J) (C/S—S-J)
Wauwatosa—Midgley, Thomas III (M)

Alaska
Adak—Jones, Austin E. (M)

Argentina
Buenos Aires—Vila, Fernando (M)

Australia
Big Bell—Champness, Peter A. (J)

British West Indies
Jamaica—Stoiser, Lee R. (J) (C/S—S-J)

Canada
Northwest Territory—Kettleson, Harold (J) (C/S—S-J)
Ontario—Zurbrigg, Homer F. (M)

Chile
Concepcion—Waissbluth, Gregorio C. (A) (R. C/S—J-A)
Rancagua—Broady, Robert G. (J) (C/S—S-J)

England
Eccles-Manchester—Glossop, Farncis G. (M)

Greece
Athens—Scalistiri, Dimitri (M)

Mexico
Chihuahua—Clayton, Richard H. (J)

Peru
Lima—del Carpio, Z. Alberto (M)
Lima—Graff, Everett H. (M)
Lima—Koehler, Henry J. (M)
Lima—Lewis, Richard W., Jr. (J)

Quebec
Val d'Or—Livingstone, David A. (M)

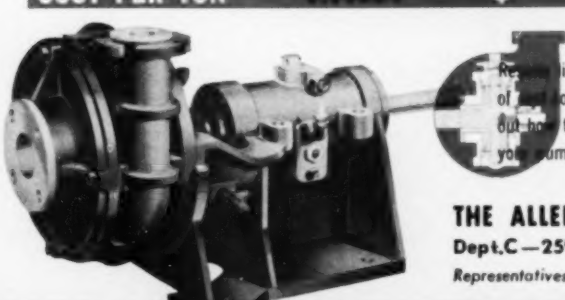
South Africa
Johannesburg—Laxen, Peter A. (J) (C/S—S-J)

Advertisers' Index

Let's look at the record

This unretouched photograph shows the Maximix Rubber Shell Liner from a Hydroseal Sand Pump after it was taken from service. Here's how it compared w[illegible] a metal pump casing of conventional design previously used in the same service:

Metal Pump Casing Conventional Sand Pump			Maximix Rubber Liner HYDROSEAL Sand Pump	
WEIGHT OF PART	1,080	POUNDS	105	90% LESS
PUMPED PER DAY	4,000	TONS	11,000	175% MORE
SERVICE LIFE	100	DAYS	340	240% LONGER
TOTAL DELIVERY	400,000	TONS	3,740,000	835% MORE
COST PER TON	0.00284	$	0.000065	98% LESS



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